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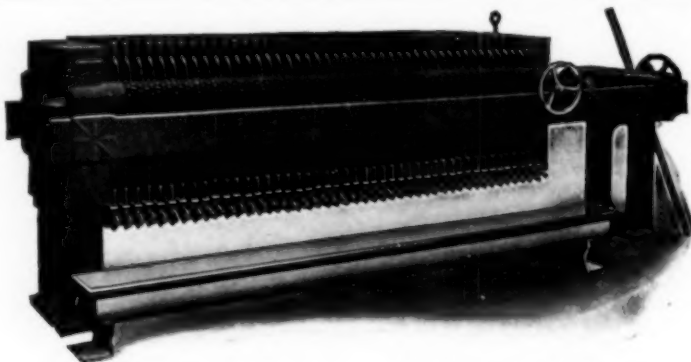
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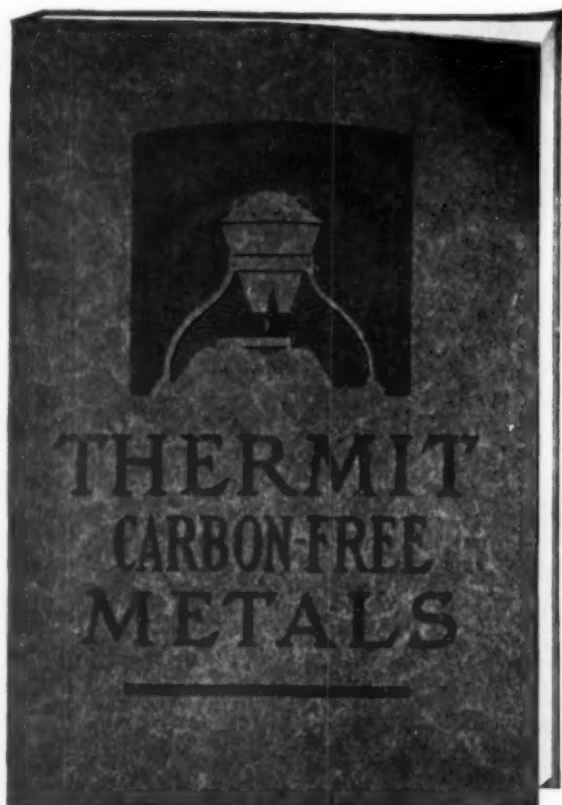
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METALLURGICAL & CHEMICAL ENGINEERING

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Synthetic Ammonia. By F. Haber and R. Le Rossignol	211
A translation in abstract of the report made by Prof. F. Haber and Dr. R. Le Rossignol to the Badische Co., which led to the development of the process on a large scale, the first factory being in course of erection in Germany. The thermodynamic principles upon which the process is based are given in detail. The construction of the apparatus is described and the results are given of the investigation concerning the best catalytic agents.	
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This article will be found to be a very valuable contribution to the scarce literature on an industry which has assumed great proportions in the last years. The author discusses the manufacture of black enameled or "black pipe" and the methods of manufacture of galvanized or "white pipe" and critically discusses the case of "black pipe" versus "white pipe" and the relative advantages of the different processes.	
The Determination of Vanadium in Ferro-Vanadium. By William W. Clark	195
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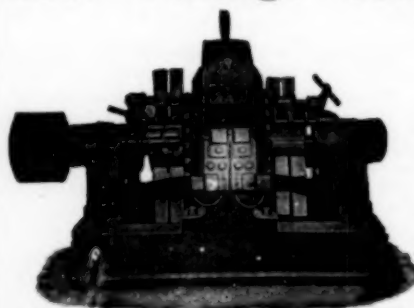
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The Haber Synthetic Ammonia Process

At last year's International Congress of Applied Chemistry Dr. Bernthsen made the important announcement that the problems involved in the Haber synthetic ammonia process have been "solved fully on a manufacturing scale and that the walls of the first factory for synthetic ammonia are already rising above the ground at Oppau, near Ludwigshafen on the Rhine." The report which Professor Haber made to the Badische Company three years ago and which led to the development of the process by this company on a large scale, has now been published and a rather full account of it is given elsewhere in this issue. It is a historic document and calls for comment on the commercial, technical, and scientific aspects of the process.

From a commercial standpoint the process has various very attractive features. As mentioned in the beginning of Prof. Haber's report, there is a large margin between the value of the ammonia and the value of the raw materials, hydrogen and nitrogen. In the form of 25 per cent ammonium sulphate the ammonia is estimated to be worth 22.25 cents per kilogram, while the cost of the nitrogen and hydrogen in it is estimated at $1.125 + 4.375 = 5.5$ cents. In view of the comparative simplicity of the process and the low power requirements it is hardly risky to say off-hand that this margin is large enough to pay manufacturing cost and a good profit.

A second attractive commercial feature is the fact that with the rapid increase in the consumption of fertilizers in recent years over-production and a drop of prices due to over-production seems a practical impossibility for years to come. Taking nitrogen fertilizers alone and taking nitrate and ammonia together, Bernthsen finds that there has been an annual *increase* in consumption of 38,000 tons of nitrogen, corresponding to nearly 185,000 tons of ammonium sulphate. Clearly enormous quantities of synthetic ammonia must be produced to affect the nitrogen fertilizer market. This explains why Dr. Bernthsen was emphasizing that the development of the new industry would not take place at the cost of other branches of industry and commerce. It may also explain why the Badische Company has withdrawn from the Norwegian electric-furnace nitrate field. Synthetic ammonia is a big enough proposition even for a very big company.

From a technical viewpoint it is interesting to compare the synthetic ammonia process with the electric-furnace processes of making nitrate and of making calcium cyanamide. The latter two employ a very high temperature and their power requirements are high. The ammonia process employs a comparatively moderate temperature—between 500 and 700 deg. C.—and its power requirements are low. From a technical viewpoint the most interesting and novel features are the use of a high atmospheric pressure and the circulation of the high-pressure nitrogen—

hydrogen gas mixture between a furnace (where the ammonia is formed) and a refrigerator (where the ammonia is liquefied and withdrawn).

Even more interesting, perhaps, is the process from a scientific viewpoint. The process will go into history as one of the biggest achievements of modern physical chemistry. Without modern physical chemistry, based on thermodynamics, the process could never have been developed. The one fundamental problem was, indeed, of a thermodynamical nature; it was to find the best temperature and the best pressure for the reaction. How keenly Professor Haber attacked the thermodynamic equilibrium problem and how systematically he solved it, is appreciated by all who know his classical *Vorlesungen über technische Gasreaktionen*. But there was another fundamental problem to be solved which is intimately connected with the first problem. This was to find the best catalyzer to get a sufficiently high reaction velocity. How the thermodynamic equilibrium and the reaction velocity influence the choice of temperature and pressure is admirably brought out in the report.

Admirable, too, is the systematic perseverant research of Professor Haber and his co-workers on the effectiveness of various catalytic agents. But the successful and clever way in which the periodic system of the elements is made use of in this research, makes the mystery of the nature of catalytic action only more mysterious and more interesting. Why should certain elements accelerate the reaction? Why should other materials (Bernthsen's "poisons") retard it? Why should the effective catalytic agents be found in certain groups of the periodic system of elements? Why should, in these groups, the elements with highest atomic weights be the best catalytic agents? There is no end of why's and apparently there is no answer. Thus, while the process emphasizes the importance of the advances made in physical chemistry in recent years, it also emphasizes the limitations of our theoretical knowledge.

Professor Haber is no stranger to American chemists. Especially among American electrochemists he has many friends and admirers. Ten years ago he attended the first Niagara Falls meeting of the American Electrochemical Society, and his able report to the Bunsen Society on the American electrochemical industries is unforgotten. In his hour of triumph his American friends sympathize with him and extend to him their hearty, sincere congratulations.

Engineers, Practical Men, and Theories

In his presidential address before the Colorado Scientific Society, Mr. George E. Collins recently made some pertinent remarks about the practical application of hypotheses and theories. Although Mr. Collins was speaking particularly of mining and not of metallurgy, it would require only a simple substitution of terms to make his comments equally pertinent to ore treatment. Comparing the so-called "practical man" with the professional engineer, Mr. Collins showed that while the former often scoffs at the ideas of the latter, regarding him as a theorist, nevertheless the practical man "also plans his work along the lines suggested by some hypothesis. The trouble is, as has been pointed out by others, there is no theorist so wild or so inveterate as your practical man; none that rides his hobby so hard or so far." This must call to the memory of every

engineer some experience with a "practical theorist"; as, for example, the man who knows that the fire assay for gold is unreliable for *his* ore; or the one who confided to us that the presence of "telluride of quick" in his ore was the principal metallurgical obstacle to its successful treatment. More serious in their consequences are the theories of subordinates in metallurgical works, who frustrate the plans and tests of their superiors because of an abiding faith in their own ideas.

Following Mr. Collins by analogy: What is needed for progress in ore treatment is the trained mind, acquainted with the literature of the subject, able to observe and to correlate and interpret results, and familiar with all the ideas of others which might have a bearing on his problem. Thus equipped, the individual will be able to form rational hypotheses on which to base experimentation. He will know when to pursue and when to abandon a line of investigation, and will be able to balance the probability of failure with the possibility of success. With all justice and consideration for the practical man, the need of the day is the trained mind.

Grinding Machines

The respective merits of the four principal types of grinding machines, rolls, and Huntington, Chilean and tube mills, have been the subject of many general discussions and some comparative tests. But despite the work that has been done in this line, the fact remains that no adequate comparative test of all the machines has ever been made, whereby the various factors of efficiency, such as power consumption, cost of operation and suitability of product, could be compared. Competitive tests in individual instances have determined the choice of machines for those particular cases, which is perhaps the best way of eliminating the less desirable machines and determining which is fittest to survive. In this way we have secured some good data, and have arrived at certain practical conclusions.

The increasing popularity of the tube mill as a secondary grinding machine, directs attention toward the deficiencies of the other types mentioned. Generally speaking, the objection to rolls is their failure to deliver a finished product in one operation, with the result that oversize in considerable quantity must be returned to the same machine. This, in turn, entails poor sizing in the trommels, and builds up feed at this point in the mill system to a tonnage which usually is considerably greater than the original feed to the mill. This condition could be avoided by a screen-faced machine with fixed maximum size of discharge, such as a Chilean mill; but a new factor would be introduced in the form of excessive slime production, which might be decidedly detrimental to the subsequent treatment of the ore. Rolls give a more granular product for concentration, the middling from which can be further ground to liberate the mineral; and for this reason rolls have been widely adopted in concentrating mills. On the other hand, the Chilean mill has been successfully used in copper concentration, for grinding an ore which must be finely divided in order to free the disseminated mineral from the gangue. But in these cases, excessive sliming has been avoided by grinding in stages, using coarse screens on the mills, and regrinding the coarser sizes of sand.

The Huntington mill works well on soft ore, but is not properly constructed to be of much effect on hard ore. It lacks the positive action of the Chilean mill or rigid rolls, and hence has little effect on an ore which is hard enough to resist crushing by the centrifugal force of the rollers against the die. On the whole, it is a rather complicated machine, which requires careful attention and the exercise of good judgment in its operation. One operator at Park City, Utah, has secured better results from this mill by connecting the upper ends of supports of opposite rollers by means of rods and a turnbuckle, thereby holding the rollers positively in position against the die.

The tube mill, and its close relative, the ball mill, are likely to displace the Huntington mill altogether, as well as make inroads on the former domain of the Chilean mill and rolls. These two machines, especially the Hardinge type, have been widely adopted as secondary grinders in concentration. They are attractive on account of their simplicity, absence of the wearing parts so common to other grinders, and the ease of replacing the grinding medium. Even for small mills the combination of a coarse crusher, ball mill and tube mill, each adjusted to produce a definite crushing effect, and thereby eliminating troublesome screens, offers such a simple and efficient means of preparing ore for different forms of treatment that it should receive attention.

The Steel Corporation's Report

The United States Steel Corporation's report for the year 1912 shows very substantial progress toward the full realization upon the improvements and extensions it has been making. The output in steel products, weighed in the form in which sold to the public, was 12,506,619 tons, an increase of no less than 1,772,624 tons over the best previous year, which was 1910. While the corporation's capacity was not fully engaged in 1912, the output was approximately only about a million tons less than capacity.

The corporation is able to meet its interest and dividend charges with very much smaller profits per ton than when it was formed, for its capacity when formed April 1, 1901, was only 8,000,000 tons, over which its actual output in 1912 was a gain of 56 per cent., while its present capacity represents a gain of between 70 and 75 per cent. Concurrent with this increase in capacity and in actual output there has been a relatively small increase in capital obligations. At the start the bonds, both of subsidiary companies and of the steel corporation itself, together with the common and preferred shares outstanding, made a total face value of \$1,400,000,000. To-day the securities outstanding amount to \$1,513,000,000. The sinking fund, interest and dividend charges have increased in hardly as great ratio. At the outset the corporation paid 4 per cent. dividends on its common stock, but assuming 5 per cent., to make a comparison with the present dividend at that rate, the sinking fund, interest and dividend charges were approximately \$85,000,000 a year; to-day they are \$91,500,000. This latter sum includes the interest paid on bonds which are carried in sinking funds, but the face value of such bonds is not included in the \$1,513,000,000 of outstanding securities already mentioned. The charges of 1901 represented \$10.60 per ton of output at capacity, while

the charges to-day represent \$6.65 per ton of output at capacity. The total earnings in 1912 represented \$9.45 per ton of actual output. The earnings covered all charges and left a slight balance for surplus, \$3,605,247.

Wall Street studies the annual reports of the Steel Corporation with much interest and pays attention to summaries of the amounts which the corporation has invested in plant betterments and extensions, for the purpose of illustrating the extent to which the corporation has "squeezed the water" out of its capitalization. The practical steel manufacturer realizes such summaries are inconclusive without adequate study also with respect to the depreciation charges which have been allowed, and on account of the vicissitudes in the steel industry it is impossible to be certain at any time how much depreciation should be allowed. While the view is apparently very superficial, it is probably much more accurate, if not so precise, to summarize the Corporation's position in this way, that it undoubtedly requires a larger capital investment to manufacture a ton of finished steel economically to-day than it did 12 years ago, when the Corporation began business, yet we find the actual showing is that in this period the Corporation has increased its capital obligations and its annual charges only 8 per cent., while it has increased its productive capacity between 70 and 75 per cent.

Since its early days the Steel Corporation has made an annual offering of its shares to employees, naming a price somewhat under the current market figure, to be paid upon the installment plan, but to bear dividends from the date of subscription. The offering of January, 1913, resulted in subscriptions by 36,119 employees, for 34,551 shares of preferred and 25,793 shares of common stock, representing respectively 0.96 and 0.49 per cent. of the total shares outstanding. Under the latest subscription, subscribing employees receive a bonus of \$5 a year on preferred shares and \$3.50 a year on common shares, as long as they retain the stock and remain in the employ of the corporation, up to a period of five years, at the end of which time the bonus which would have inured to those who dropped out is divided among the survivors. It would be interesting to know the situation at present as to participants in previous distributions, how much stock is retained by the original subscribers, the classes respectively whose bonuses have expired and whose bonuses continue. An accurate statement as to the former would, of course, be impossible, as shares sold are not always transferred on the books.

The Corporation's years of largest output were 1910, with 10,733,995 tons, and 1912, with 12,506,619 tons, showing a gain for 1912 of 16.5 per cent. over the record. There were great changes in the distribution of output, for some products actually lost in tonnage, while others made great gains. Rails lost more than 12 per cent. In minor railroad products there was a large increase in axles, but almost as large a loss in steel car wheels. The most important gain was in sheets and tin plates, 39 per cent., while tubular goods showed an increase of 28 per cent. On the whole it is readily observed that the Corporation carried its tonnage to a higher degree of finish, on the average, in 1912 than in 1910, so that the physical value of its output increased more than the 16.5 per cent. which the tonnage increase represented.

Readers' Views and Comments

Simple Approximate Rules for Temperature Conversions

To the Editor of Metallurgical & Chemical Engineering:

Sir.—The following simple, easily remembered, approximate rules for mentally converting high temperatures in Fahrenheit degrees into one in Centigrade degrees and the reverse, may be of interest to those dealing with furnaces; the troublesome factor 5/9 is thereby evaded. Whether they are already generally known or not in this form, the writer does not know.

Double the Centigrade temperature and deduct one-tenth of that figure, (or deduct 10 per cent, then double it). Thus 1000 deg. C. doubled is 2000, less a tenth (200) leaves 1800 deg. Fahr. For all ranges the error is that this value is always 32 deg. too small; hence it may be made accurate by adding 32.

Halve the Fahrenheit temperature and add one-tenth of that figure (or add 10 per cent, then halve it). Thus, 2000 deg. Fahr. halved is 1000, plus a tenth (100) gives 1100 deg. C. Accurately it is 1093.3+; hence the approximate value is only about 1/2 of 1 per cent too high. Again 3000 deg. Fahr. halved is 1500, plus a tenth (150) gives 1650 deg. C. Accurately it is 1648.9—; hence the approximate value is less than a tenth of 1 per cent too high. For the lower temperatures the errors become greater.

If 32 is first deducted from the Fahrenheit values, then the values given by the latter rule are always almost exactly 1 per cent too low for all ranges, hence could be made nearly absolutely correct if desired by adding 1 per cent.

These rules are easily remembered as it is well known that the Fahrenheit values are *not quite* double the Centigrade ones, hence something must be *deducted* after doubling, the Centigrade values, or must be *added* after halving the Fahrenheit values; and that this correction factor is 10 per cent is easily remembered as this is the simplest possible of all factors, being merely a shift of the decimal point.

The mathematical proofs of these rules are very simple, as the correct factors are 5/9 and its reciprocal, beside the 32.

Philadelphia, Pa.

CARL HERING.

Determination of Oxygen in Iron and Steel.

To the Editor of Metallurgical and Chemical Engineering:

Sir.—In the February number of METALLURGICAL AND CHEMICAL ENGINEERING appears an article by Mr. R. H. McMillen, on the application of the electric resistance furnace to the determination of oxygen in iron and steel.

The writer's¹ modification of the Ledebur method (*Jour. Ind. & Eng. Chem.*, vol. 3, p. 372), is referred to, and the double electric furnace as well as certain other modifications are proposed as improvements.

The laboratories under the writer's supervision have had probably a larger and wider experience with the determination of oxygen in iron and steel than any other in this country. In our opinion the modifications proposed by McMillen present no improvements over the process used by us, and are open to criticism in respect to several important points. In a routine laboratory where a large number of oxygen determinations have to be made, the time element is of great importance. With the gas blast furnace using a silica tube it is possible to cool down immediately by turning on the cold air blast, so that accurate results can be obtained in one-half hour. The double electric furnace modification runs the test for a full hour, and if the analysis is started with a cold furnace, it will require an hour to reach the specified temperature of 950 deg. C., whereas the gas blast furnace can be brought to temperature in a few minutes. In our practice we often exceed this temperature and have never found that we obtain higher results with the higher temperature.

McMillen proposes the use of a porcelain boat instead of the nickel boat used by us. We especially avoided the use of porcelain, owing to the tendency of that material to adsorb moisture from the air.

The most important criticism of McMillen's modification that we have to offer is with respect to weighing the phosphoric acid absorption tube filled with hydrogen. We believe this to be bad practice, and numerous tests made in this manner have shown discordant results owing to the tendency of hydrogen to diffuse through "air-tight" stoppers.

The McMillen method is almost identical with the one published by the writer, with the exception of the points discussed above, and for the reasons stated it does not appear that any contribution has been made to the older method. McMillen again points out the well-known fact that the Ledebur method will not determine total oxygen, owing to the fact that certain oxides and silicates are not reducible by hydrogen. Walker and Patrick have proposed a method for determination of total oxygen, by making use of a vacuum electric furnace of the Arsem type. (Paper before Eighth Intern. Congr. of Appl. Chem., *Met. & Chem. Eng'g*, vol. X, p. 629.) The installation for this method is very expensive and at best it could hardly become a routine determination in any chemical laboratory. On the other hand, it should not be forgotten that in most cases it is only the determination of oxygen combined with iron that yields the data desired, and it is this information which can be readily obtained by the Ledebur method. The amount of oxygen combined with silica and other oxides unreducible by hydrogen has no bearing on the question as to whether iron or steel is properly deoxidized in the process of manufacture.

ALLERTON S. CUSHMAN.

Institute of Industrial Research,
Washington, D. C.

* * *

To the Editor of Metallurgical & Chemical Engineering:

Sir.—In the criticism of Dr. Allerton S. Cushman upon my article, "An Application of the Electric Resistance Furnace to the Determination of Oxygen in Iron and Steel," published in your February issue, he objects to the use of the electric resistance furnaces in my modification on account of the time required to bring the furnaces to a full heat and to cool them after the combustion is completed, thereby curtailing the number of daily determinations.

I consider that the advantages offered by this method more than compensate for the time lost, when the convenience of this means of heating is taken into account, and also the little attention the furnaces require. After the operation is started and until the combustion is completed, the operator is practically free to attend to other duties, so that very little of his time is counted against the oxygen determination. When the number of daily determinations to be made requires it, more furnaces can readily be added, using the one preheating furnace for all.

Dr. Cushman next objects to the use of the porcelain boat on account of the moisture absorbing tendency of this material. In making use of the porcelain boat this property was not lost sight of. No opportunity should be given for it to absorb any appreciable amount of moisture, as the dried boat need not be exposed to the moisture of the laboratory but a very short time while receiving the sample and being placed into the tube. Since the empty boat, which has been exposed to the air under conditions similar to those in the actual determination, is used in running the blank, a correction for the moisture which may be added this way is made when the blank is subtracted.

In the Cushman method (*Jour. Ind. & Eng. Chem.*, Vol. 3, p. 372) the statement is made that either platinum or silica

boats are used, and no reference whatever is made to the nickel boat which he now states is used.

Dr. Cushman's third and most important criticism is to the practice of weighing the phosphoric acid absorption tube filled with hydrogen. I consider the displacement of hydrogen by dry air as in the Cushman method unnecessary, since in drying the hydrogen filled absorption tube the same time is given and the same conditions maintained before weighing both before and after the combustion.

At various times experiments were conducted in order to determine whether or not the diffusion of hydrogen through the stoppers makes any variations in the weights which would materially affect the results. A phosphoric acid absorption tube was filled with dry hydrogen and dried over concentrated sulphuric acid for fifteen minutes before weighing, this procedure being repeated many times, and in no case was there any appreciable variation in weight. A second weight was made thirty minutes after the first in each case, and in no instance did the weight vary more than 0.0002 gram from the first, showing that, in a reasonable time limit, no appreciable error is introduced from this source.

R. H. McMILLEN.

Crescent Laboratory, Crucible Steel Co., of America,
Pittsburgh, Pa.

Dr. Struthers' Reply

To the Editor of Metallurgical & Chemical Engineering:

Sir:—Enclosed please find a copy of a letter to me from Mr. Charles F. Rand, president of the American Institute of Mining Engineers, which is a conclusive reply to the direct and inferred criticisms, so far as they refer to me, appearing in the letter of Messrs. Corning and Stone in your issue of September, 1912.

I respectfully request, as a matter of justice, that you will publish this present communication and the letter from Mr. Rand in an early issue of your paper.

JOSEPH STRUTHERS.

New York City.

Mr. Rand's letter is as follows:

"Dr. Joseph Struthers, New York.

"Dear Sir:—Your letters of February 21 and 24 have been received, together with copies of the METALLURGICAL AND CHEMICAL ENGINEERING of September, 1912, the *Mining and Scientific Press* of January 25 and February 8, 1913, and the *Engineering and Mining Journal* of February 8, 1913, in which latter the suggestion is made that you demand an investigation; with the request on your part that the criticisms appearing in these publications as far as they relate to you should be considered by the Board of Directors or the Executive Committee of the Institute.

"Your letters and the accompanying newspapers have been carefully considered by the Executive Committee.

"We are very glad to state that no charges of any kind against you are in existence before the Board.

"Your accounts as Assistant Treasurer have been audited annually by a firm of certified public accountants and always found correct.

"As to the excursions which are specifically referred to by the *Mining and Scientific Press*, you are entitled to the statement which we very gladly make that you did not originate the system by which such excursions were conducted. The system had prevailed for many years, and before you became connected with the administrative work of the Institute.

"With regard to the reference in the *Mining and Scientific Press* to the New Haven and Spokane meetings, we now find that there was no excursion connected with the New Haven meeting, and understand that you had no financial relation beyond the receipt of salary and expenses to the excursion connected with the Spokane meeting. Both of these meetings were held while you were Assistant Secretary and before you became Secretary of the Institute.

"A committee of the Board and Council has already inquired

into the conduct of the excursion to Japan which was under your charge, and exonerated you, as shown in a report which has already been published.

"As to the letter of Messrs. Corning and Stone, published in METALLURGICAL AND CHEMICAL ENGINEERING last September, we appreciate the position taken by Dr. Raymond and you at the urgency of your friends, to abstain from any newspaper controversy which might have been harmful to the Institute. At the same time we are pleased to say, concerning the general charge which might be construed to include you, that you have not received an excessive salary and that you have not been inefficient in the office work.

"The Executive Committee consists of Charles F. Rand, Benj. B. Thayer, A. R. Ledoux, Joseph W. Richards, and James F. Kemp.

"Yours very truly,

"(Signed) CHARLES F. RAND,

"President American Institute of Mining Engineers."

Early American Mining Schools

To the Editor of Chemical & Metallurgical Engineering:

Sir:—On page 120 of the March number of your journal is a note on mining schools. In this the name of the School of Mining and Practical Geology of Harvard University is omitted. This school was founded in 1865 by the appointment of Prof. J. D. Whitney to the chair of geology in 1865. In 1866 Prof. R. Pumpelly was appointed to the chair of mining, and in 1871 William H. Pettee, who had been acting as instructor, was appointed Assistant Professor of Mining.

The first class of four members was graduated in 1870. Two of these Gannett and Marvin, entered the United States Geological Survey, where Gannett still remains. In 1869 the class was taken to Michigan by Professor Pumpelly to see the iron and copper mines, and afterwards J. D. Whitney took them to the mountains of Colorado. During this part of the trip Mt. Harvard and Mt. Yale were ascended and named.

This party, so far as I know, was the first college party to make a summer visit to the Rocky Mountains. The party consisted of Prof. J. D. Whitney, of Harvard; Prof. William Brewer, of Yale; Instructors Pettee and Sharples, of Harvard, and four students, William M. Davis, who has succeeded Professor Whitney as geologist at Harvard; Joseph Bridge, Henry Gannett, now chief geographer of the United States, and A. Marvin, who was on the United States Survey, but died in 1876. Mr. Bowles, son of the editor of the *Springfield Republican*, was also with the party. They were also accompanied by Charles Frederick Hoffman as topographical engineer. The school was discontinued in 1875, being merged with the Lawrence Scientific School.

Boston, Mass.

S. P. SHARPLES.

The Solubility of Alumina in a Bath of Fused Fluorides.

To the Editor of Metallurgical & Chemical Engineering:

Sir:—In the March number of your journal, page 121, Dr. Charles A. Doremus discloses from a paper published by St. Claire Deville in 1857, that alumina like silica was known to be soluble in a mixture of potassium and sodium fluorides fused in an externally heated crucible; and that while Deville succeeded in electrolytically depositing silicium by feeding silica into such a bath, he failed to deposit aluminium under the same conditions. Deville's conclusion was that alumina unlike silica, is not decomposed by the alkali metals.

Dr. Doremus stated that had Deville's proportions been more propitious the alumina should have been decomposed by the electric current in the same manner as silica. Deville's bath contained no aluminium fluoride. This latter deduction made by Dr. Doremus would be natural if he considers alumina when fused an electrolyte whose molecules are directly decomposed without a secondary reaction. The old literature of electrolysis teaches that pure oxides like water are not directly decomposed by the current. Deville was expecting that the liberation of

potassium or sodium on the cathode would react on alumina in the bath with the formation of aluminium, while at the same time oxygen or carbonic-oxide would form against this carbon anode and fluorine would have an opportunity to combine with the alkali metal of the alkali oxide that would be forming in the bath. He knew that an alkali metal would reduce silica.

Dr. Paul Héroult in his patents and the experts for Mr. Charles M. Hall in litigation that was never decided all held that alumina in itself was capable of being directly decomposed by electrolysis. In the latter case the experts for the Cowles Electric Smelting & Aluminum Company held that it was well known that alumina was soluble in cryolite, but that alumina was not a direct electrolyte.

The following experiment performed by the writer about twenty years ago must have some scientific bearing on this subject:

A round hard carbon block, 3 in. in diameter and about 3 in. in height, was taken. Three holes were bored into one of the flat surfaces of this block to a depth of about an inch and of nearly the same diameter. One of these holes was filled with pure sodium fluoride NaF , another was filled with cryolite $(\text{NaF})_2\text{Al}_2\text{F}_6$, and the third was filled with cryolite containing a little added aluminium fluoride. The block was now uniformly heated until the material in all three holes came to quiet fusion. Three small pieces of aluminium, all having the same weight, had been previously prepared each about the size of a small bean. One of these pieces was now dropped into each of the three fused baths. The bath of sodium fluoride gave off a great abundance of sodium vapor which burned with its characteristic yellow flame. The bath formed of cryolite gave off some yellow flashes of sodium vapor. The bath containing a slight excess of aluminium fluoride, above that of cryolite, gave off no sodium vapor. Upon allowing the block to cool and breaking it, it was found that all the aluminium had been consumed with the formation of aluminium fluoride in the bath formed of sodium fluoride. On weighing the button from the cryolite bath, it was found to have lost a very small per cent of its weight, while the bath containing a slight excess of aluminium fluoride gave up a button having its original weight.

Dr. Percy in the year 1855, and Rose independently in 1856 discovered that cryolite could be directly reduced and aluminium produced by using sodium as a reducing agent, but they found the efficiency very low. This was due to too great a quantity of sodium fluoride accumulating in virtue of the reduction of the aluminium fluoride with sodium.

All three baths were heated to the same temperature in the above described experiment, which was about the temperature employed in the present commercial manufacture of aluminium. It will be noted that when aluminium fluoride is in excess to that contained in cryolite $(\text{NaF})_2\text{Al}_2\text{F}_6$, aluminium does not reduce sodium fluoride, and on the other hand, when sodium fluoride is in excess, aluminium does reduce sodium fluoride, and sodium acts in just the reverse manner as to aluminium fluoride in the composition.

Déville might have said at his time that aluminium and sodium reverse their positions in the electropositive and negative series in these different mixtures, but modern chemists, I think, would explain this as a phenomenon of mass action, but it still leaves open the question as to whether alumina in a pure condition is or is not like water a non-electrolyte.

ALFRED H. COWLES.

Sewaren, N. J.

New Mill at Manhattan, Nevada

The amalgamation mill of the Big Four company, at Manhattan, Nevada, was put in operation early in March. While this is a small mill, it is well equipped with modern machinery, and its operation is regarded as an important event in the history of Manhattan. The ore is free-milling. It is first crushed in a gyratory crusher, then in stamps. The stamp product is classified into sand and slime by Akins classifiers, the sand being reground in tube mills. Amalgamation follows tube milling. The capacity will be 120 tons per day.

Atlantic City Meeting of American Electrochemical Society

The spring meeting of the American Electrochemical Society will be held in Atlantic City from April 3 to 5 (Thursday to Saturday of the first week of April). Headquarters will be at the Hotel Traymore, and the professional sessions for the reading of papers will be held at the Solarium at the hotel.

Thursday and Saturday will be completely devoted to reading and discussion of papers, while on Friday, April 4, an excursion will be made to Philadelphia, leaving Atlantic City on the Pennsylvania Railroad in a special car attached to the 7:45 a. m. train and arriving at Broad Street station at 9:10 o'clock. The works of the Crucible Steel Castings Company at Lansdowne, Pa., will be visited in the morning where the making of crucible steel will be seen and a 2-ton Roechling-Rodenhauer induction steel furnace will be inspected.

At noon the party will return to Philadelphia and a visit will be paid to the University of Pennsylvania, where Dr. Edgar F. Smith, the provost of the University, will address the society.

After lunch visits will be paid to the Harrison Bros. Company on the Grays Ferry Road (leads, paint, lithophone, contact sulphuric acid) and to the United Gas Company Works at Point Breeze (water gas, illuminating gas, and gas testing laboratories).

The party will return to Atlantic City, leaving Broad Street station at 4:14, arriving at Atlantic City at 5:39 p. m. On the evening of Friday, after the lecture by Professor Kenrick, a smoker will be held at a place to be announced at the meeting.

The program of papers is as follows:

Thursday, April 3, 1913

10:30 a. m. Annual business meeting of the society, in the Solarium, of the Hotel Traymore; reports of the Board of Directors; announcement of the annual election; miscellaneous business.

The following papers will then be read and discussed:

Conduction and radiation of heat, by *J. Langmuir*.

Experiments with furnace electrodes, by *F. A. J. Fitzgerald* and *A. T. Hinckley*.

Aluminium nitride, by *J. W. Richards*.

Some tests of the Edison Storage Battery, by *C. W. Bennett* and *H. N. Gilbert*.

Concentration cells containing organic liquids immiscible with water, by *R. Beutner*.

Concentration changes in Copper Sulphate Electrolysis, by *C. W. Bennett* and *C. O. Brown*.

The 3-phase, 2-phase induction furnace, by *A. E. Greene*.

Making Electric Steel without Slag, by *A. E. Greene*.

On Friday, April 4, 1913, a lecture will be given in the Solarium, Hotel Traymore, by Prof. Frank B. Kenrick, of Toronto, on the subject of hyperbasis.

Saturday, April 5, 1913

10:30 a. m. Presidential address by President *W. Lash Miller*, introducing the symposium on the electrodeposition of metals.

Symposium on Electrodeposition of Metals:

Electrodeposition of Gold and Silver, by *Francis C. Frary*.

Electrodeposition of Copper, by *C. W. Bennett*.

Electrodeposition of Brass and Bronze, by *C. W. Bennett*.

2:30 p. m.

Electrodeposition of Cobalt and Nickel, by *O. P. Watts*.

Electrodeposition of Lead, by *F. C. Mathers*.

Solid thick lead cathodes from lead lactate solutions, by *F. C. Mathers* and *W. B. Cockrum*.

Registration will begin on the evening of Wednesday, April 3, in the lobby of the Hotel Traymore, and a meeting of the Board of Directors will be held on the same evening at 8:30 p. m.

Dr. W. Lash Miller of the University of Toronto, is the president of the American Electrochemical Society. Dr. J. W. Richards of Lehigh University, South Bethlehem, Pa., is the secretary.

Annual Meeting of the British Institute of Metals

Abstracts of Papers Presented

The annual general meeting of the (British) Institute of Metals was held in London on March 11 and 12.

The report of the Council states that the year 1912 has marked a period of material development in the activities of the Institute. The number of members is 614.

Corrosion Committee

From the report of the corrosion committee we note that considerable progress has been made in the investigations of the causes of corrosion of tubes of the four types of alloy selected by the committee, namely, 70:30 brass, Admiralty mixture, lead-bearing brass, and Muntz metal. Twelve tubes of each alloy have been tested in the special condenser plant for nine months, *i. e.*, from April to December, 1912. Three tubes of each composition have been withdrawn for detailed examination; this, however, has not yet been completed. A preliminary examination has shown that a small amount of corrosion such as that usually met with in tubes that have failed in practice, has occurred in some of these tubes, but has not penetrated to any considerable depth. All the tubes have been found to be covered with a scale of composition similar to that typical of those used in the mercantile marine. Investigation on these tubes is still proceeding.

The plant itself was closed down temporarily on Dec. 31, 1912, pending the supply of further funds for working expenses which amount to about £100 per annum. It is highly desirable that the experiments with this plant should be continued at the earliest opportunity, and this will be done as soon as the necessary funds are forthcoming.

An extensive scheme of laboratory experiments has been devised with the object of elucidating the nature of reactions which take place during the processes of corrosion and scale formation. It has been found necessary to continue a number of experiments for periods of several months, as the collection of data is necessarily slow. Hitherto it has not been possible to devise any satisfactory "acceleration" test, so work in this direction has been abandoned for the present.

A considerable number of badly corroded tubes have been sent in for examination by various shipping firms and manufacturers, and afforded useful information. The investigator is now desirous of obtaining a number of tubes that have endured exceptionally long service in marine or land condensers.

In the following we give abstracts of the papers presented at the meeting:

Metallic Filament Lamps

A paper on "Metal Filament Lamps," by Mr. **Alexander Siemens**, summarizes the history of the use of metallic filaments in glowlamps.

Owing to the uneconomical working of carbon-filament lamps, endeavors were made to replace carbon by one of the rarer metals, which can sustain a higher temperature than carbon without disintegrating.

Auer was the first to utilize osmium, prepared by a squirting process, and his methods were followed by several other inventors experimenting with various metals.

The first lamp to have actually drawn metal wire as its filament was the tantalum lamp manufactured by Siemens & Halske, Berlin. They also succeeded in drawing an alloy of tungsten and nickel, but before that process was perfected the General Electric Company, of Schenectady, patented a process to make pure tungsten ductile, which is described in the paper.

Corrosion of Condensed Tubes

A paper by Mr. **Arnold Philips** deals with "The Corrosion of Distilling Condenser Tubes."

Main and auxiliary condenser tubes become corroded on the sea water side; corrosion on the steam side of the tubes practically never occurs.

In distilling condensers the reverse of this is the case. No

corrosion occurs as a rule on the sea water side of the tubes but frequently occurs on the surfaces exposed to the steam.

The absence of corrosion of the sea water side in distilling condensers is apparently due to the facts that (1) the sea water flows outside the tubes, which tubes are generally placed vertically; on this account no particles of carbon, ash, etc., can find a lodgment on the surface of the tubes on the sea water side and thus what the author has shown to be one of the most important causes of corrosion is absent. (2) The tubes are usually in direct metallic connection with the casing, being expanded directly into the tube plates; this permits the full galvanic protective effect of any protector bars and of the casing itself when this is wholly or partially of iron or steel.

The corrosion on the steam side of distilling condenser tubes is due to hydrochloric acid in the steam.

The origin of the hydrochloric acid is the dissociation of salts contained in sea water, probably magnesium chloride. This dissociation only takes place to any noticeable extent when the primary steam tubes in the evaporators project above the surface of the brine which is being boiled.

The same distilling condenser used with two evaporators, one of which has steam coils projecting above the brine surface and the other having drowned steam coils, produces distilled water containing hydrochloric acid in the former case, but free from this acid in the latter case.

The effect of the acid-containing steam from the evaporator is to corrode the surface of the distilling condenser tubes and the distilled water obtained contains not only free acid but also chlorides of copper, zinc, and (if tin-coated tubes are used) also tin.

The presence of copper in the distilled water is particularly objectionable both for potable uses and also because when passing into the boilers it tends to set up further corrosion and pitting on the steel surfaces.

The cure to avoid these troubles is to use distiller evaporators which contain completely drowned coils. A palliative to prevent copper passing into the boilers with the distilled make-up water, if this contains copper due to a faulty form of evaporator being used, is to pass the distilled water through a zinc scrubber which removes the copper.

Chemical methods for detecting and measuring minute amounts of free hydrochloric acid in the distilled water obtained from distilling condensers are described.

Corrosion of Aluminium

A paper by Dr. **G. H. Bailey** deals essentially with the action of water and salt solutions on aluminium.

In the course of the investigation the effects of variation in the nature of the waters used and in the concentrations of the salt solutions have been examined. A large number of results are given illustrating also the dependence of corrosion on temperature and on the quality of the aluminium employed.

The author also gives a method by which reliable measurements of the rate of corrosion may be made, and from an examination of the experimental evidence he draws the following conclusions:

That aluminium of high purity is less readily acted upon than that of lower purity and that the presence of sodium and copper in the metal increase the rapidity of corrosion. Well-annealed metal is also more resistant to corrosion than unannealed metal. He finds that in general the corrosion of aluminium is a process of oxidation, and that as a matter of fact metal exposed for several months to water or salt solutions from which the dissolved air has been expelled underwent no corrosion whatever. The normal course of corrosion (excluding the action of acids and alkalis) is thus a transformation of aluminium into alumina, which separates out as a flocculent precipitate without any of the aluminium passing into solution.

Microstructure of German Silver

A paper by Mr. **O. F. Hudson** deals with the microstructure of German silver. Samples of German silver which had been annealed for varying lengths of time at a temperature of

7000 deg. C. were examined microscopically. The alloys which consist of a single solid solution, require many hours annealing at this temperature to bring them to a condition of true equilibrium when the solid solution appears to be perfectly homogeneous. German silvers when annealed thus commonly possess a characteristic "cored" structure which is independent of the normal twinned crystalline structure also seen.

The rather coarsely crystalline structure which results from the long annealing necessary to remove the "cores" is not in itself a sign of deterioration in rolling qualities. The results of tests on samples annealed for various times at temperatures up to 1000 deg. C. showed that, under laboratory conditions, the rolling qualities of good German silver are not impaired by severe annealing.

Hardness tests, using the scleroscope, showed that prolonged annealing, accompanied by pronounced crystal growth, does not lead to any decrease in hardness beyond that due to the normal annealing operation.

Effect of Rapid Cooling on Constitution

A paper by Mr. G. H. Gulliver deals with "the quantitative effect of rapid cooling upon the constitution of binary alloys."

The author shows how to calculate the proportions of the constituents of a rapidly cooled binary alloy by means of the information obtained in the usual manner from very slowly cooled alloys. The importance of the subject is due to the fact that the constitution of a cast alloy cooled at ordinary rates lies between that of the very slowly cooled and that of the very quickly cooled mixture, so that its limits of variation with change in the rate of cooling can be now specified.

The method of calculation is applied to the lead-rich alloys of the lead-tin series. The proportion of eutectic in a just solid alloy, the proportion of liquid and of solid in an alloy at a temperature between its freezing and melting points, and the rate of solidification of an alloy at a given temperature, when slowly and when rapidly cooled, are found numerically. The apparent form of solidus obtained from a rapidly cooled alloy is given, together with the variation in the apparent position of the saturation point which accompanies variation in the rate of cooling. It is shown that the position of the apparent saturation point depends not only upon the rate of cooling, but still more upon the composition of the alloy, and further that the proportion of eutectic present in a rapidly cooled alloy is sensibly independent of the curvature of liquidus or solidus.

The paper is accompanied by a number of diagrams which clearly show the differences between the conditions which prevail during slow and rapid cooling respectively.

Heat Treatment of Gun Metal

A paper by Messrs. H. S. Primrose and J. S. Glen Primrose deals with the "practical heat treatment of Admiralty gun metal."

As difficulties often arise in producing gun-metal castings to fulfil specified physical tests and withstand hydraulic pressure, the authors describe their experimental work with the object of finding a simple and reliable method of improving this commonly used industrial alloy so as to obtain the highest possible strength and elongation accompanied by greater soundness and homogeneity of the metal. They found that even in the absence of blowholes, which constitute the commonest source of unsoundness, cast gun metal behaves unsatisfactorily under hydraulic tests due to the presence of microscopical pores formed between two constituents of widely different properties.

A series of tests was carried out to investigate the practical value of various kinds of heat treatment, such as reheating, quenching, and annealing test bars of gun metal conforming to the Admiralty specification of 88-10-2, which had been cast in sand and chill molds and then cooled at different rates. The sets of normal and treated bars were tested physically and examined microscopically to determine exactly the effect of quenching from different temperatures and also of annealing for various periods of time at different temperatures.

The first set of tests made showed no improvement was

effected by reheating and quenching the bars, as the strength and elongation fell off rapidly as the temperature was raised. This result was unexpected as it is known that pure bronze is increased in strength by similar treatment.

More satisfactory results were obtained by simply annealing the metal at various temperatures, followed by moderately slow cooling, as this was found to considerably increase both the tensile strength and the elongation. The most marked improvement was found to be produced by annealing the bars for half an hour at a temperature of 700 deg. C., as the physical tests showed lower results both above and below this critical point. The time factor was also investigated and found to be critical for the $\frac{5}{8}$ in. thickness of metal used at thirty minutes, although only a slight improvement resulted after annealing for twenty minutes, and a slight diminution in strength occurred if annealing was too long prolonged.

The practical applications were discussed and examples quoted from practice in which material not otherwise defective failed under the hydraulic test due to water sweating through the microscopic pores, but after suitable annealing conformed to the most rigid tests. The increased homogeneity of the metal after the heat treatment was shown to be due to the absence of eutectic segregations from the microstructure, leaving only the strongly interlocking crystals of alpha solid solution of the tin and zinc in the copper. It was anticipated that this heat treatment would minimize corrosion troubles.

The headquarters of the Institute of Metals are at the Caxton House, Westminster, S. W., London. Mr. G. Shaw Scott is the secretary.

The Non-Ferrous Metal Market

No marked changes have occurred since our last report. Business has been fairly quiet, and the quotations were not realized on all transactions. At the latest reports the different markets have assumed a firmer tone, and it is expected that this condition will continue.

Copper.—The domestic market has been quiet and few purchases have been made. Most of the business has been effected at concessions under the quoted price. The bulk of the sales have been made to foreign buyers who have been attracted by the prevailing low prices. Little Lake Copper has been sold, and the market has not yet been firmly established on a 15-cent basis. The last quotations are 14 $\frac{3}{4}$ @15 cents for Lake, and 14.70@14.80 cents for electrolytic.

Tin.—Wide fluctuations have been noticeable in the London market, but have not been reflected in this market. The favorable February statistics gave a firmer tone to the March market, and liberal purchases were made, especially for future delivery. The latest quotation is about 48 $\frac{3}{4}$ cents, New York.

Lead.—A firmer undertone has prevailed in this market and sales have been heavier. The market is active at 4.17 $\frac{1}{2}$ @4.20 cents, St. Louis, and 4.32 $\frac{1}{2}$ @4.35 cents, New York.

Spelter.—Prices have advanced a little under the effect of a better inquiry for metal. Quotations have a wide range according to delivery, the last being 6.05@6.15 cents, St. Louis, and 6.20@6.30 cents, New York.

Other Metals.—Aluminium has shown a firmer tone since our last report, being quoted at 26@27 cents, New York. No material change is noted in the antimony market, prices ranging from 8 $\frac{1}{8}$ to 9 $\frac{1}{2}$ cents for various brands. The quicksilver market is quiet; the New York quotations being \$40 per flask of 75 lb. The San Francisco price is the same, except for export, the latter being \$37.50.

Nissen stamps have become popular machines for crushing ore in Rhodesia, where three new mills are being equipped with these machines. Roasting also is a feature in several of the new Rhodesian mills.

Tilting Crucible Melting Furnaces for aluminium, brass, bronze, copper, gold, silver and other non-ferrous metals are the subject of illustrated catalog 44 of the W. S. Rockwell Company, of New York City.

Evolution of Methods of Handling Slime—I

By H. N. Spicer

Modern methods of handling ore-slime, either in concentration or cyanidation, are comparatively recent developments in the history of metallurgy. It is necessary only to review the literature on this subject for the past decade to find that slime has been the source of much trouble, and that its successful treatment has been the subject of many investigations and inventions. The result is a comparatively high state of develop-



FIG. 1.—REMOVING SAND FROM COLLECTORS AT NUNDYDROOG MINE

ment in the art of handling this once troublesome material, and the successful extraction of valuable metals from it.

The methods employed in classifying, thickening, agitating and filtering slime probably are matters of common knowledge to well informed engineers; but it may be surprising to many to learn of the crude yet profitable methods employed in those countries where the latest mechanical appliances are not yet in use. The writer visited last year some of the important mining districts of the world, and being specially interested in the slime problem, took many notes and interesting photographs on the prevailing methods in the different countries. In India, as in many other foreign countries where nature labor is employed, the methods of operation may appear crude to the uninitiated; but it must be borne in mind that these methods have been determined by the prevailing labor conditions, and that those in authority deserve much credit for determining the economic limit to which this class of labor can be used.

Methods in Vogue in India

India at this time presents a particularly interesting phase of the slime problem, because the industry is in the midst of a period of transition from the old to the new. Prevailing methods of handling slime probably are the crudest to be seen anywhere in the world; and yet it will be but a short time until they are wholly abandoned in favor of modern methods. Already there is some evidence that the old order of things has

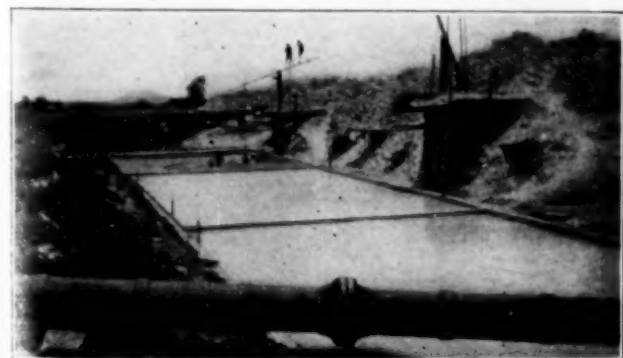


FIG. 2.—REMOVAL OF SLIME FROM COLLECTING TANK AT NUNDYDROOG MINE

passed, and that the strange operations shown in some of the accompanying illustrations will no longer be seen.

Gold mining has been a profitable industry in India for many years, but it is only within the last year that the first slime-treatment plant was erected. Prior to that time some sand had been leached with cyanide solution, but only such slime was treated as might remain with the sand and still give a

leachable product. The consequence is that large tonnages of slime await treatment at the different mines. It is probable also that a large portion of the leached sand is sufficiently valuable to be reground in tube mills and treated as slime.

The principal reason why so little attention has been paid to cyanidation in India is that the gold recovery by amalgamation has been so efficient. In the Mysore district, for example, from 82 per cent to 85 per cent of the gold in the ore is recovered by stamp-amalgamation. The sand and slime represent a value worth recovering, however, and they are being saved with that end in view; but now that cyanidation is about to be widely adopted, the slime problem is complicated by the fact that the ore is crushed in water, necessitating the dewatering of both sand and slime before further treatment.

The mill at the Balaghat mine, on the Kolar gold field, Mysore, is a very old one. The pulp from the stamps runs into concrete pits in the ground; the slime and water overflow, and when the pit is full of sand it is excavated by hand and stacked for future treatment. The slime flows to other concrete tanks, where the pulp is thickened by free settling to about 50 per cent moisture and then excavated and stacked by hand.

Sand and Slime at Nundydroog

Methods of dewatering at the Nundydroog mine represent a step in advance of those at the Balaghat; they are shown in

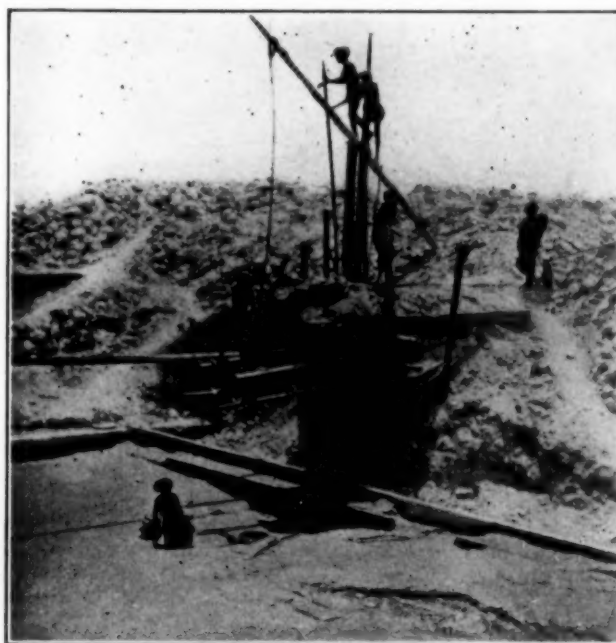


FIG. 3.—ELEVATING SLIME FROM THICKENING TANKS, NUNDYDROOG

Figs. 1, 2, 3 and 4. The stamp-mill pulp flows to steel collecting-tanks, which are equipped with Butters distributors, as shown in Fig. 1. When a tank has been filled, the flow of pulp is diverted and the sand allowed to drain. It is then dug out by hand, the work being done for the most part by women using a kind of hoe and a small wicker basket. Stacks of old tailings may be seen in the background.

The slime overflowing these sand-collecting tanks passes through launders to a series of concrete settling basins, shown in Fig. 2. Here the slime settles to a consistency of about 55 per cent solids, after which it is elevated and stacked by means of a native contrivance known as a "deckool." Its operation will be better understood by reference to Figs. 2, 3 and 4. In Fig. 2 the bucket is shown lowered to the woman in the slime pond, who fills it, after which it is elevated as shown in Fig. 3 to a man who dumps it into a launder, Fig. 4. As most of the slime is too thick to run in any launder, it is pushed or scraped along to the dumping place by women provided with implements for the purpose.

The Ooregaum 80-stamp mill is shown in Fig. 5. At this

point reference may be made to the construction of mill and other mine buildings in the Mysore district, which are built of dressed granite, with walls 3 ft. thick. To one unfamiliar with local conditions, this must appear to be an extravagance; but as a matter of fact, the cost of this type of construction is less than for frame and corrugated iron.

The stamp batteries are arranged in two sets, facing each



FIG. 4.—DUMPING SLIME INTO LAUNDER, NUNDYDROOG MINE

other, and the combined pulp flows out at one end of the mill. Formerly the sand was allowed to settle in pits with walls of masonry, as shown in Fig. 6, the slime overflowing to settling basins. Later this was abandoned and the pulp conveyed in an open launder to a series of steel sand collecting-tanks, shown in the middle background of Fig. 7. At the head of the open launder are two spitzluten, by means of which from 10 per cent to 12 per cent of the coarsest sand is eliminated for future

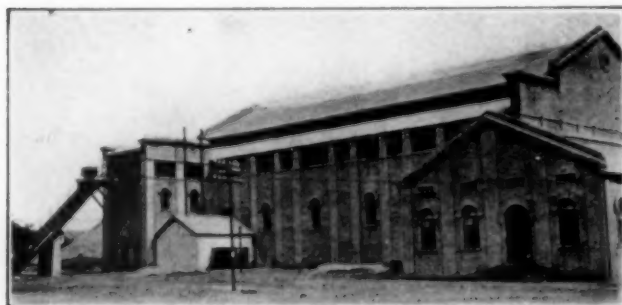


FIG. 5.—OOREGAUM 80-STAMP MILL

regrinding. It falls to the ground and is taken to a dump by women, who fill the cars as shown in Fig. 6.

The remainder of the sand and slime passes on, as above indicated, to steel sand-collectors, Fig. 7, the slime overflowing to the concrete slime pits in the foreground. These pits have pyramidal bottoms, and are approximately 35 ft. square at the top, and 16 ft. deep in the center. Formerly the thickened slime

was pumped to the dump, but since the installation of a 30-leaf Butters filter the procedure is as follows: The slime is allowed to settle as much as possible and the supernatant water is pumped off by means of a portable pump shown on a truck at the right of the settlers. Cyanide solution is now added in quantity to give a pulp containing about two parts of solution to one of solids. Compressed air is turned into the pulp from the pipes shown extending from the left side to the center of the settlers. This agitation keeps the solids in suspension while the pulp is being pumped to a series of four 19 ft. by 10

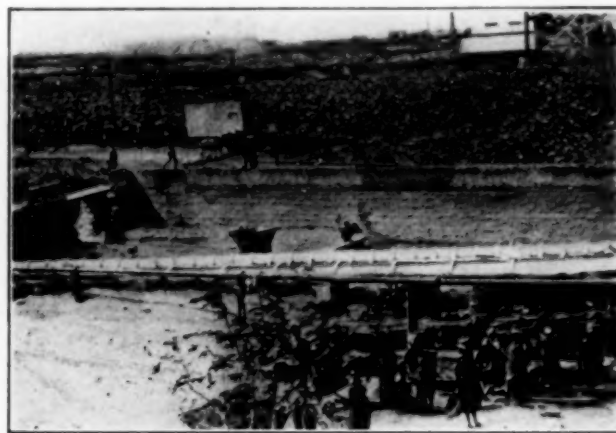


FIG. 6.—ELIMINATION OF THE COARSE SAND AT OOREGAUM MILL

ft. Pauchuca agitators. After agitation the pulp is filtered in a Butters filter, shown in Fig. 8. This illustration is particularly interesting as affording a view of the first vacuum filtration plant erected in India. The filter builds a cake of about $1/16$ in.; the cycle of filtration, washing and discharge occupies from 50 to 60 minutes.

Sand and Slime Separation at the Mysore Mine

At the Mysore gold mine, sand and slime are separated by the system shown in Fig. 9. The stamp mills are out of view at the right. The battery pulp flows through the launder shown supported on a trestle at the farther end of the row of tanks, and thence into two transverse launders built of masonry. Pipes leading from these transverse launders deliver the pulp



FIG. 7.—THICKENING SLIME AT OOREGAUM MILL

to Butters distributors placed over two rows of filter-bottom steel tanks. Only one row of tanks is shown in the view, the other being at the left. Each tank is provided with an overflow launder for slime, which passes to a series of concrete settling basins where it is handled in the same manner as at Nundydroog, as shown in Figs. 2, 3 and 4.

When a tank has been filled with sand, the feed is discontinued and the charge drained as dry as possible by vacuum applied under the filter-bottom. Naturally this system of separating sand and slime is not the most efficient, particularly in the le-

ginning of the operation when the tank is empty. The consequence is that the lower part of the charge contains more or less slime which affects the uniformity of drainage and leaves a variable quantity of moisture in the sand. After draining the charge as dry as possible it is discharged by hand through doors in the bottom of the tank, falling onto belt conveyors, which transport it to the dump shown in the back-



FIG. 8.—PACHUCA AGITATORS AND BUTTERS FILTER AT OOREGAUM MILL

ground. Extensive experiments are now being carried out to determine the best type of cyanide plant to build for the treatment of this ore.

First Complete Slime Plant in India

The first complete slime-treatment plant in India has been built at the Champion Reef. Fig. 10 shows the Champion Reef 180-stamp mill, with slime settlers and Pachuca tanks at the right. The battery pulp flows over plates and then to cone classifiers, the slimy overflow from which gravitates to two 50 ft. by 8 ft. slime-collecting tanks, which have 4-ft. conical bottoms, and in which the slime is thickened to a consistency of about 55 per cent solids. After decanting the water the slime is sluiced out with cyanide solution, and pumped to three 33 ft. by 10 ft. Pachuca tanks connected in series. Vacuum filtration completes the treatment.

Transition to Modern Methods

Turning now from past operations to present and future

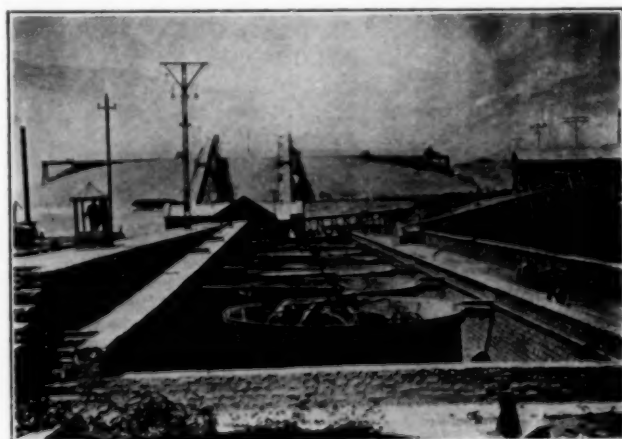


FIG. 9.—SAND COLLECTORS AND CONVEYORS AT MYSORE GOLD MINE

plans, we find many instances of the transition to modern methods of handling slime. At Nundydroog a slime plant was designed last year and is now being erected. It comprises eight Dorr duplex classifiers, four tube mills, three 30 ft. by 10 ft. Dorr thickeners, four 38 ft. by 10 ft. Pachuca agitators and a 120-leaf Butters filter. At Ooregaum the old clumsy method

of thickening the slime is to be replaced by Dorr thickeners. At Mysore experimental work is under way to ascertain what proportion of the ore should be treated as slime in an up-to-date plant. In all probability tube mills will be used, as well as the latest developments in mechanical classifiers, thickeners, and vacuum filters.

At the Jubtil mines of Anantapur, in the Madras Presidency, 60 miles north of the Kolar Field, a 20-stamp mill is about completed which will be the first in India to adopt crushing in cyanide solution. This will greatly simplify the slime problem by eliminating the necessity for dewatering. The equipment includes stamps, amalgamating-plates, tube mills, Dorr thickeners, Pachuca agitating-tanks and Butters filter.

Near Raichur, in the native state of Hyderabad in the Deccan, is an old gold mine known as the Hutti (Nizam). It has



FIG. 10.—CHAMPION REEF 180-STAMP MILL PACHUCA—AGITATORS AT RIGHT

been operated intermittently for many years, the sand and slime being handled in as crude a way as in the old plants of the Mysore district. A modern mill is now being erected, to treat about 70 tons of ore per day, using a tube mill, Dorr thickener, and small Butters filter.

Throughout India the process of evolution in gold metallurgy, particularly in the treatment of sand and slime by cyanidation, is very noticeable, and within a year or two the old ideas and methods will have been completely abandoned.

Denver, Colo.

Advantages of Small High Speed Electric Furnaces.

By Carl Hering

The question is often asked how much electric energy will be required to melt a specified amount of metal in a foundry for casting purposes. It is not generally understood, however, that to answer this question intelligently and to the satisfaction of the proposed user, there are other factors which one should also know as they may have an important bearing on the result. For instance, a small furnace which melts many small charges per day, and a large one which melts only a few large charges per day, can have the same daily output, and yet may require quite different amounts of electrical energy, hence different amounts per pound. The amount of energy required may also be materially different depending upon whether a given quantity of metal is melted rapidly or slowly.

The purpose of the present article is to analyze and discuss these two features of electrical furnaces, to show their significance and importance, and to illustrate the results with some estimated approximate quantitative data.

The user of a commercial furnace is not interested in theoretical efficiencies or in the so-called radiation losses, electrode losses, etc.; the efficiency which he is interested in is the amount of product produced per dollar; this may be represented by the simple formula $P/\$$, in which P is the amount of the product produced, and $\$$ is the total cost of producing that amount; the greater this "efficiency" the better he is pleased. Hence the duty of the engineer is to do everything and anything to increase this commercial efficiency, even though this may include some things which by themselves might appear to be bad practice because they are inefficient; if they increase this commercial efficiency they become good practice.

Some years ago the writer made the statement that the most efficient electric furnace is the electric fuse for protecting electric circuits when it "blows out" on a short circuit. Its efficiency, as a furnace, for sudden short-circuit currents, is probably high in the nineties, that is, very nearly perfect. The reason for this is that the heat is generated so rapidly that the metal is melted, and the intended process is completed, before any appreciable amount of the heat has had time to get lost.

This little "furnace," therefore, teaches us the importance of proceeding as quickly as possible not only in generating the heat, but also in doing whatever is to be done with the metal while it is melted, such as casting it, purifying it, mixing alloys, etc.; in other words, *to reduce as much as practicable the time during which each pound of metal must be kept hot*. During the whole of that time it is losing heat, and this "stand-by loss" increases in proportion to this time; hence if the melted metal can be melted and handled twice as quickly, this loss will be approximately halved. And as these stand-by losses are by no means small and insignificant, their reduction becomes important.

There are in general two ways in which these "time" losses in electrical furnaces can be reduced in practice. The first is to get the heat into the metal as quickly as practicable, that is, to force the furnace to the greatest practicable extent. This means to increase the power delivered to the furnace (the kilowatts, not the kilowatt hours) as much as practicable, and to get this energy into the metal as quickly as possible; in those resistance furnaces in which the heat is set free in the metal itself, absolute perfection has been reached in not losing any time in getting the heat into the metal after the heat has been generated, as it gets there instantaneously; in arc and other furnaces depending chiefly upon radiation and conduction it will necessarily take some time for the heat to get disseminated in the bath, and it is difficult to see how that time can be reduced except by very high temperatures and by rapid circulation. Furnaces which can be forced are, therefore, better in this respect than those which cannot.

The second way to reduce these stand-by "time" losses is to reduce the quantity of metal which forms one charge, to the smallest amount that the particular conditions warrant, so that the heated charge may be disposed of as quickly as practicable, thereby reducing the time that each pound of metal is losing heat. For instance, if the purpose is to cast into molder's flasks taking, say, 50 pounds each, then the ideal furnace would be one which melted a charge of about 50 pounds at a time; the losses in such a furnace would be less than in one which contained a charge of 100 pounds, as the latter has to hold the charge twice as long. Hence the size of the charge must be decided by the user, but his attention should be called to the possible economy of the smaller charges, as it would probably not suggest itself to the average foundryman; in fact, he would probably have difficulty in understanding it even after it was explained to him, as the best electric furnace practice differs from the customary fuel furnace practice.

In some cases it is important to have large charges, as, for instance, when an alloy of very definite proportions is required; it may be too troublesome to mix small batches, especially when samples have to be analyzed. Also when slag must be changed. A certain amount of time may also be necessary in certain refining operations. In all such cases the conditions govern what should be done. But even in such cases one should bear the above in mind and make the charges as small and the time of heating as rapid as the particular conditions permit, if economy of energy and the reduction in the size of the furnace are important.

By thus reducing the charges to the smallest which are practicable, and melting as quickly as possible, the size of the furnace may be greatly reduced, so much so that it may be possible in some cases to make it portable either on rails or suspended from an overhead track, thus enabling it to be brought to the flasks or other molds in a foundry, thereby permitting casting directly from the furnace into the molds; this not only

saves some of the superheat, probably a very appreciable amount, but it also saves labor, pouring ladles, heating the latter, etc. It may also make it practicable to enclose the whole furnace in a closed tank for heating in a vacuum or under pressure, for refining.

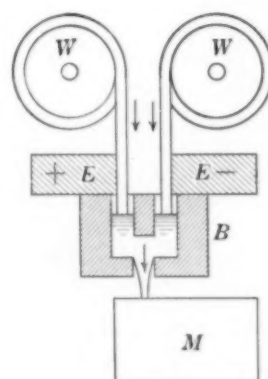
The ideal plant for a foundry, for instance, where regular work is done, would be a furnace which contained only about as many pounds of metal as are cast into one flask, and which melted this much metal in the time it takes to move the furnace from one flask to another and to do the pouring. For small brass castings, such a furnace would be small enough to be readily handled on a track or suspended from a trolley rail, and could be tilted so as to cast directly into the flasks. Such a furnace is now under construction for a brass foundry, though it was not considered wise to go to the extreme of the ideal; in the present case, for instance, the furnace will contain enough metal for several flasks.

It is by making use of every possible advantage of the properties of the electric furnace in ways like these that the greater cost of the source of heat may be more than overbalanced by other gains of money value.

Elsewhere the writer has shown¹ that the losses through the walls are in general greater per pound of metal, for small than for large furnaces, and that, therefore, furnaces should better be built of large capacity; yet the arguments given above seem to lead to opposite conclusions; the facts are that both conclusions are right when properly interpreted (see Figs. 1, 2 and 3 in the concluding summary).

In general small machines are not as efficient as large ones, and yet from what was said above the contrary is true about furnaces. Again, a machine which is forced or overloaded is in general running less efficiently, and yet the contrary is here claimed to be the case for furnaces.

These apparent inconsistencies are due to the fact that with furnaces there are so many different factors involved which affect the result in different and sometimes contrary ways, that the problem sometimes becomes quite confusing. The only satisfactory way to study them is to determine the laws governing each



SKETCH OF IMAGINARY FURNACE

factor alone and then in various combinations. The writer has published at various times analyses and discussions of the laws of the losses through the walls, and those through the electrodes; in the present article the effect which the element "time," or speed of operation, has upon the losses is discussed with the other two sources of losses.

When problems become complicated because there are so many variable factors, it is often instructive and helpful to study extreme cases. In the present instance time or speed is the chief element under discussion; it appears that the greater the operating "speed" of a furnace the more economical it will be; imagine therefore a furnace in which this factor is carried to its extreme limit. Such an imaginary furnace is shown in outline in the adjoining sketch. Let B be a small block of refractory material with a very small U-shaped hole through it, and with the electrodes EE attached to a source of current. Let M be the mold into which the melted metal is to be cast and let WW be reels of wire of that metal capable of being fed rapidly into the holes. The current through the metal in the holes heats and melts it, and the melted metal immediately runs into the mold. Suppose the wires to be fed in as fast as the melted metal can flow out, and suppose the electric current supplies energy fast enough to do the melting at this rate and that the "pinch effect" did not interfere.

¹Heat Conductances Through Walls of Furnaces. Trans. Amer. Electrochem. Soc., Vol. 14, p. 225 (1908).

It will be seen that this would be an extremely small and highly efficient furnace; it would probably have the highest possible efficiency of any electric furnace, the reason being that the metal is melted and discharged into the mold so rapidly, and the quantity of melted metal in the furnace and therefore its surface, is so small, that there would be very little time or opportunity for the heat to get lost. This furnace is based on the principle of the electric fuse which, as described above, is the most efficient form of electric furnace.

This imaginary extreme case points out the directions in which practical furnaces should tend to be developed, namely, rapidity of melting and smallness of charge.

In order to get some idea of the possible economy gained by applying the above principles, a series of furnaces for melting brass have been calculated and the results are given in the accompanying tables and curves. Actual figures have here been assumed for the melting energy and for the rate of losses, the total being equal to their sum. But as these basic data are not yet known to the desired degree of precision these actual figures are only roughly approximate and no great weight should be placed on them; it is to the *relative* and not to the *absolute* results that attention is here called; on the other hand, these relative results would probably not be greatly different for more accurate basic data.

The basic data which have been assumed here are as follows: When there are no losses of heat it is assumed to take about 8.25 kwh. to melt 100 lb. of brass, including about 10 per cent. more energy to superheat it. This figure is not yet known definitely; it may be and probably is somewhat less; it is, moreover, different for different brasses; if too high it is on the safe side, hence is conservative. The weight of brass is taken to be about 540 lb. per cu. ft. It is thought that the loss through the walls may for brass temperatures be taken as about 1 kwh. per sq. ft. of surface of the charge;² this is also only approximate, but is probably not far wrong; it would probably require careful designing to bring the loss down to this figure. The electrode loss is assumed to be 20 per cent. of the total; this is believed to be liberal. The hearth is assumed to be

TABLE I.

SAME CHARGES—DIFFERENT RATES OF MELTING

Note.—Brass. Volume of charge 3.70 cu. ft. Diameter of hemispherical hearth 2.42 ft. Loss from charge 1 kw. per sq. ft. Electrode loss 20%. Theoretical consumption for melting and 10% superheating of 100 lbs. brass, 8.25 kwh.

Charge in Lbs.	Time of Melting in Hours	Rate of Melting in Pounds per Hour	Kilowatt Hours for Melting	Kilowatts for Melting	Loss Through Walls in Kilowatts	Loss in Electrode in Kilowatts	Total Power in Kilowatts	Efficiency in Percentage	Kilowatt-hours per 100 Pounds
2000	4	500	165	41.3	13.7	14	69	60	13.8
2000	3	667	165	55.0	13.7	17	86	64	12.9
2000	2	1000	165	82.5	13.7	24	120	69	12.0
2000	1	2000	165	165	13.7	45	224	74	11.2
2000	0.5	4000	165	330	13.7	86	430	77	10.7
2000	0.25	8000	165	660	13.7	168	842	79	10.5
2000	0*	∞	165	∞	0	0	∞	100	8.25

*Imaginary theoretically perfect furnace in which there is instantaneous melting, like in an electrical protecting fuse.

hemispherical in order to reduce its surface, and therefore the wall loss, to the least possible. In most furnaces these losses would probably be greater, hence the relative differences pointed out below would in most cases be still greater.

In Table I and Plate I, the data are given for a series of furnaces for melting brass each of which has the same charge, namely, one short ton of 2000 lb., hence all have the same size of hearth, but in each of them the charge is melted at a different rate; the slowest rate, 500 lb. per hour, would be considered slow in a fuel furnace of that size, while the most rapid rate, 8000 lb. per hour, would probably be quite impossible in a one ton fuel furnace. The last one in the table is an imaginary theoretically perfect furnace, given merely for the purpose of showing the absolutely extreme case.

²Possible Reduction of the Power Consumption in Electric Steel Refining Furnaces. Met. & Chem. Engineering, November, 1911, p. 590.

The total required power in kilowatts is made up of that required for melting, plus that lost through the walls (a constant in this case), plus that lost through the electrodes, these being the only two unavoidable losses. These component parts are shown by dotted lines in the plate, the total sum of all three being shown by a thick line.

It will be seen that as the speed of melting is increased the power in kilowatts, of course, becomes greater and quite rapidly so at the higher speeds, but the losses become less in proportion, hence the economy becomes considerably better for the higher speeds, as is seen by the efficiencies and by the kilowatt-hours required per 100 lb.; the latter is the crucial figure of interest to the user as it represents his power bill. It will be noticed from the curves that in this case the latter varies by a straight

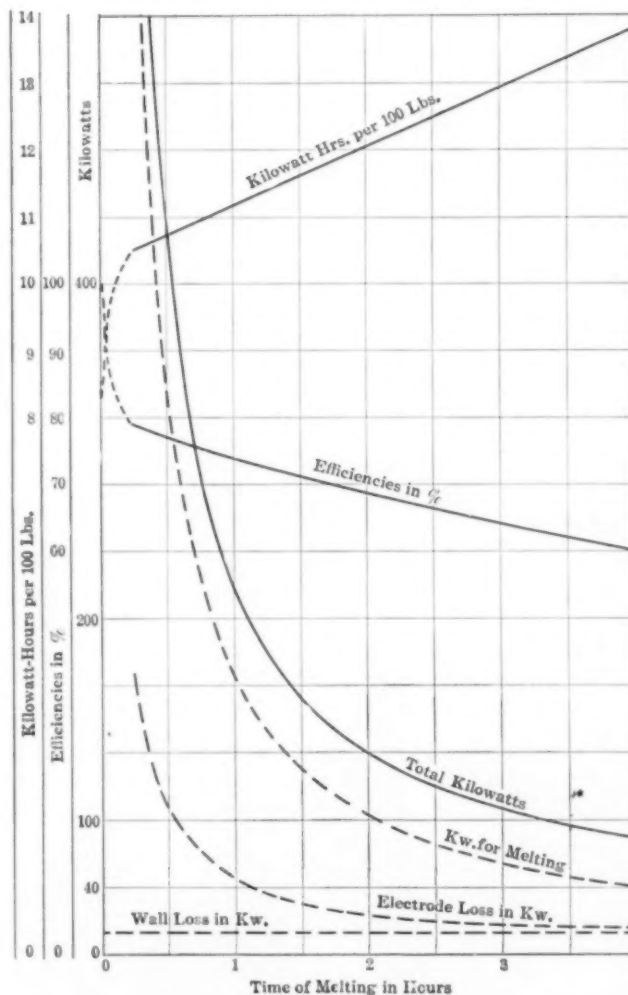


PLATE I.—RESULTS FOR A SERIES OF BRASS FURNACES WITH THE SAME CHARGE, BUT OPERATED AT DIFFERENT RATES OF MELTING

line relation. It will also be seen that the reason why the efficiency is so poor for the slowest speed is because the losses approach in amount to the power required for melting.

Table I and Plate I therefore show that, having a furnace of a given capacity of hearth, it pays well to force it as much as other conditions warrant. When the user generates his own power, greater kilowatts (greater speed of melting) means a greater cost of generating plant, which therefore tends to offset the advantages, hence the usual "happy medium" must be determined.

Comparing a slow rate of 500 lb. per hour with one of 4000 lb. it will be seen that although the latter is eight times as fast, the power is not increased in that proportion, 69 kw. compared with 430, but is only about 6¼ times as great. The efficiencies are 60 and 77 per cent., hence considerably more favorable for the higher rate. The power bills will be as 13.8 to 1.07 for the same quantity of metal; hence almost 30 per cent.

greater per pound for the slower rate. Hence the melting capacity of the plant may be increased eight fold with only about $6\frac{1}{4}$ times as much power, and the cost of power per pound of metal will be much less. Large foundries could, therefore, be operated cheaper than small ones, and the latter might to advantage do their casting in the shortest practicable time. Another illustration is given in Figs. 3 and 4 in the concluding summary.

Having now shown from Table I and Plate I, that with a given furnace of any stated capacity of hearth, it pays well to force it as much as conditions permit, the next feature to be discussed is the question of the amount of the charge, hence the size of the furnace, as the "ideal" furnace indicates that the smaller it is the better. This may sometimes be decided by other considerations than economy of power, as for instance by the weight of a desired large ingot or casting for which the charge is intended, and in such a case the only rule is to make the charge as small as one of these ingots or castings and not large enough for several, provided this is not impracticable. But assuming that one is otherwise free to choose the size of the charge an illustration of the relative results is given in Table II and Plate II.

In this second set it is assumed that the rate of melting has been fixed, and it is desired to know whether this can be done best with many smaller charges or with a few large ones. Here again the data has been estimated for a series of different furnaces, besides the imaginary perfect one. It is here assumed that the required rate is 500 lb. per hour. The data then shows the losses, power, efficiency, etc., of furnaces of different sizes, any of which will melt at this same rate of 500 lb. per hour. The same basic data is used as before, and the hearth is again assumed to be hemispherical. In the curves the kilowatts for melting, for supplying the losses through the walls, and in the electrodes, are shown by dotted lines, and their sum, the total power, is shown by a heavy line.

As the melting is now assumed to be done at the same rate in all, in pounds per hour, the power used (the kilowatts, not the kilowatt hours) for melting is the same for all, and its curve is therefore a horizontal line. The loss through the walls (the total, not that per pound of metal) is now the factor which increases rapidly as the furnace gets larger; this is due to the fact that the metal is subjected to this loss for a greater length of time.

It will be seen from the curves how greatly the efficiency improves as the furnaces are made smaller, and how fast the

TABLE II.
DIFFERENT CHARGES.—SAME RATE OF MELTING

NOTE.—BRASS. Rate 500 lbs. per hour. Hemispherical hearth. Loss from charge 1 kw. per sq. ft. Electrode loss 20%. Theoretical consumption for melting and 10% superheating of 100 lbs. brass, 8.25 kwh.

Charge in Pounds	Time of Melting in Hours	Volume in Cubic Feet	Diameter of Hearth in Feet	Kilowatts for Melting	Loss Thru Walls in Kilowatts	Loss in Electrodes in Kilowatts	Total Power in Kilowatts	Efficiency in Per centage	Kilowatt-hours per 100 Pound
2000	4	3.70	2.42	41.3	13.7	13.8	68.8	60	13.8
1500	3	2.78	2.20	41.3	11.4	13.2	65.8	63	13.2
1000	2	1.85	1.92	41.3	8.7	12.5	62.4	66	12.5
500	1	.926	1.52	41.3	5.5	11.7	58.4	71	11.7
250	0.5	.463	1.21	41.3	3.5	11.2	55.9	74	11.2
125	0.25	.231	.96	41.3	2.2	10.9	54.3	76	10.9
*	*	0	0	41.3	0	0	41.3	100	8.25

*Imaginary theoretically perfect furnace of infinitely small size, in which 2,000 lbs. are melted in 4 hours with continuous feeding and instantaneous melting.

kilowatt-hours per 100 lb. diminish as the furnaces are reduced in size; this is important to the user because that factor represents the power bill. (The curve for the latter happens to coincide with that for the kilowatts due to the scales here used; in all cases, however, the two quantities will be proportional under the assumptions here made.)

The conclusion to be drawn from Table II and Plate II is that for the same rate of melting, the smaller the furnace the more

economical it will be in power consumption; besides this the first cost will also be less. Hence the rule is to make the furnace as small as the desired charge will permit. If a certain ingot is desired make the charge equal to that one ingot and not to two or more. If the metal is to be cast into molds, make it equal to that for one mold; but as this may not always be practicable, then make it equivalent to as few molds as are.

As a concrete example taken from Table II, a 2000 lb. furnace may here be compared with a much smaller one for 125 lb., hence one-sixteenth. The large one will require about 69 kw. for 4 hours, the smaller one will require only about 54 kw. for the same 4 hours, and will deliver in that time sixteen charges of 125 lb. each, one every quarter of an hour, the rate of melting being the same in both. Hence there will be a saving of

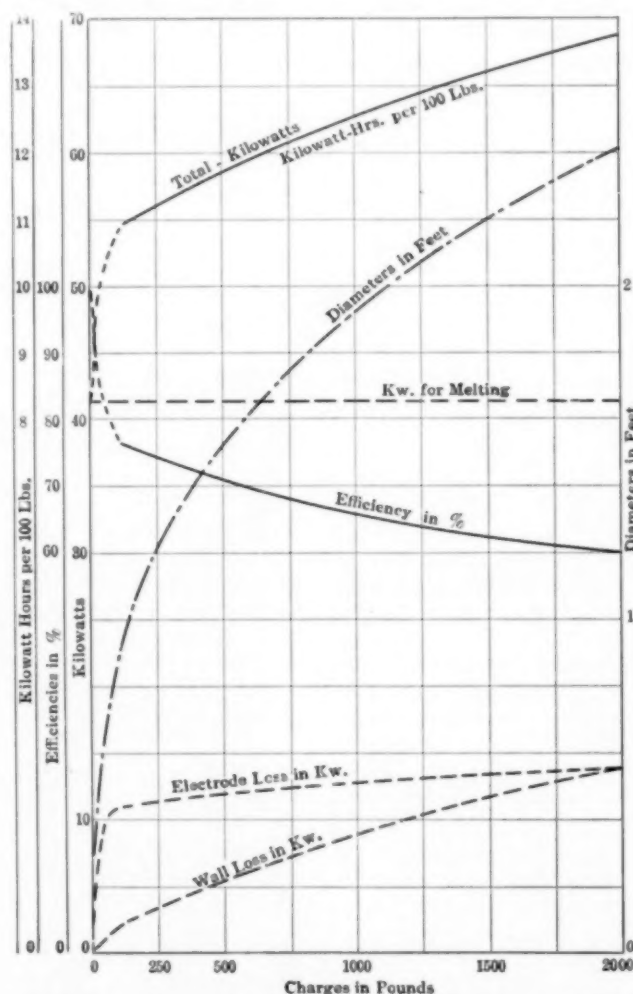


PLATE 2—RESULTS FOR A SERIES OF BRASS FURNACES OPERATED WITH THE SAME RATE, BUT WITH DIFFERENT CHARGES

about 15 kw. in favor of the smaller one. The energy consumption, and therefore the power bill, will be at the rate of 13.8 kwh. per 100 lb. for the large one, as compared with 10.9 for the small one, or about 27 per cent. better, in favor of the smaller one. Another comparison is given in Figs. 2 and 3 in the concluding summary.

As stated above, there will often be other conditions which will determine the limit to which one can go in increasing the rate of melting and decreasing the charge, and therefore the size of the furnace. The above discussion merely shows the directions in which one should tend to go in designing a furnace, other conditions permitting.

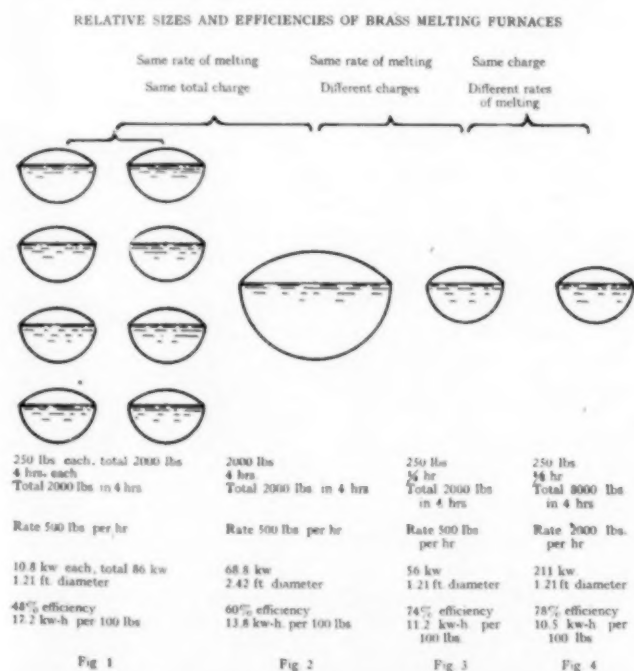
There are also some other minor factors which, for the sake of simplicity in an analysis which is already somewhat involved, are not considered here and which in some cases may affect the conclusion more or less; their importance should be examined in each case. It is true, for instance, that in a large slow-speed furnace the temperature is not at the maximum at the start,

hence the wall losses will at first be less. The extra labor of filling and emptying a small furnace frequently instead of a large one less frequently, should also be considered. On the other hand the very appreciable reduction in first cost of a smaller furnace for the same daily output, has not been considered here; nor the advantages of a reserve unit which may become practicable when the furnaces are small and cheap. Also the saving of labor, superheat, crucibles, their preheating, etc., when the furnace becomes small enough to be transported to the molds. The engineer must, of course, consider these and many other factors in his final decision.

The academician may find entertainment in expressing the above deductions in algebraic form. The writer believes that a series of concrete examples like those in these tables and curves, gives a clearer insight to the relations than abstract figures and letters do. Moreover, algebraic analyses are often burdened with complicated factors which in practice turn out to be negligible quantities.

Summary.—The general conclusions of this analytical investigation may be briefly summarized as follows:

The total consumption of energy per pound of metal, and



FIGS. 1 TO 4.—RELATIVE SIZES AND EFFICIENCIES OF BRASS MELTING FURNACES

therefore the power bill, will decrease quite appreciably: (a) the more rapidly the melting is done, and (b) the smaller the charge, and therefore the furnace; both might be combined by saying: the less the time that each pound of metal is held in the furnace. This is based on the energy required for melting and the losses in the walls and electrodes. Such reductions in the size of the furnace also diminish its first cost and may in some cases make it portable so that it may be taken to the molds; the ideal melting furnace for small castings is one which melts as fast as it can be moved from mold to mold, and poured; for small brass castings such a design is not impracticable; the same may be true for small steel castings also.

No great importance should be attached to the approximate actual figures for brass melting furnaces given in the tables, although they are probably not far wrong; attention is here intended to be called chiefly to the relative values.

Pictorially these deductions concerning the relative values may be represented by the following diagrams Figs. 1-4, which are based in part on the tables and are drawn to scale. Another feature which was brought out in a previous paper is here included because it at first sight appears to be contradictory, but in fact is not.

Figs. 1 and 2 show that it is much more economical in power and in plant cost to use one large furnace instead of eight smaller ones of the same combined capacity of metal *when all the smaller ones are being operated at the same time*; this was described in a previous paper. Fig. 1 represents usual foundry practice in that a large number of small furnaces is used, all operating together. The rate of melting here assumed would be considered a moderately slow rate for fuel furnaces.

Figs. 2 and 3 (illustrating Table II and Plate II) show that it is much more economical in power and in plant cost to melt smaller charges at a time and to melt them in succession in the same furnace, though at the same rate of melting; the furnace will then also be considerably smaller. The smaller furnace of Fig. 3, though melting no faster than the large one, namely, 500 lb. per hour, will have the same daily output and will be more economical in power consumption.

In electrical parlance, Figs. 1 and 3 show that it is far more economical both in power and plant cost to melt the charges "in series" than "in multiple."

Figs. 3 and 4 (illustrating Table I and Plate I) show that it is much more economical in power to melt a given charge quickly, four times as rapidly in this particular case. The same is shown to a more striking degree by comparing the furnace of Fig. 3 with one of the eight furnaces of Fig. 1. The rate of melting assumed in Fig. 4 is about fifteen to twenty times as fast as the usual rate with fuel furnaces of that small size, and is approximately the rate used in a large 20-ton reverberatory furnace.

Comparing the two extreme cases with each other, namely, Fig. 1 and Fig. 4, it will be seen that Fig. 1 represents in general the usual foundry practice of to-day in so far as a large number of small furnaces are operated together; while Fig. 4 represents what would seem to be the best practice with electrical furnaces, namely, a single small furnace melting rapidly a large number of small charges in succession. A comparison of the data of Figs. 1 and 4 will give an idea of the very marked increase in economy obtained by the small, quick-acting furnace.

The furnaces of Figs. 1, 2 and 3 all have the same daily output, while that of Fig. 4 has four times as great an output per day, although it is no larger than that of Fig. 3.

Philadelphia, Pa.

Reform of Procedure in Patent Suits by the New Rules

By Dyer Smith

It is perhaps not fully known among manufacturers and inventors that a quite new method of procedure in the trial of suits for the infringement of patents went into effect Feb. 1. This change results from the new rules of practice for the Courts of Equity of the United States, promulgated by the United States Supreme Court.

New Rules of Practice for the Courts of Equity, Promulgated by the Supreme Court

The new practice will undoubtedly, in most cases, cause greater expedition and lessened expense to litigants, because of a general simplification of procedure, removal of causes for delay, and shortening up of times within which action may be taken. Especially, it is believed, will these beneficial results spring from the requirements of the rules that testimony shall be taken, in all usual cases, orally in open court, before the judge.

Under this new practice, the testimony is taken in one continuous session, in court, followed by the arguments of the lawyers, and the parties and the judge give their attention entirely to the matter at hand until its conclusion.

The rules now in force take the place of those originally promulgated in 1822 and 1842 and not changed greatly since the latter date. The present revision is the work of a committee of Supreme Court justices, aided by committees of lawyers residing in the nine circuits into which the United States

is divided. The changes which have gone into effect as the results of their labors are considerable, and, while some of the innovations will have to be "worked out," the new practice is clearly a greater boon to the owners of patents.

Under the old practice testimony was almost always taken out of court, in patent suits, and reduced to writing to be read by the judge after the hearing by him of the lawyers' arguments. It usually ran to a considerable length, and was generally more expensive and unsatisfactory, it is believed, than the testimony in open court of the new practice. The former methods were a continual cause of complaint to inventors and manufacturers, so that it was commonly said that a poor litigant had no chance in a patent suit against one well supplied with funds. The aim of the new rules is to remedy this condition as far as possible.

The first point to be noted in the new rules is that the technical forms of pleadings in equity, generally speaking, are abolished. Demurrers and pleas, always strongholds of delay, are done away with, the defenses formerly presented by these forms of pleading now being made in a simple way by motion to dismiss the bill or being incorporated in the answer. The answer of the defendant now merely sets out in short and simple terms the defense to each claim asserted by the bill brought by the plaintiff against him, avoiding general denials, but specifically admitting or denying or explaining the facts upon which the complaining party relies in his bill.

The answer must also, briefly and simply, state any counter-claims or set-off which the defendant may have, arising out of the same transaction as that upon which the bill of complainant was brought, or which might be the subject of an independent suit in equity against the opposing party, thus permitting various matters to be settled in one suit which might otherwise require the filing of cross-bills or bringing of separate suits by the defendant against the plaintiff. The bill of complaint, according to the provisions of the new rules, is to be framed in as simple and concise a manner as possible.

Not only are the pleadings simplified, but the times within which they can be filed are considerably shortened. The subpoena issued when the bill of complaint is filed is returnable in twenty days, to the office of the clerk of the court, and the defendant is required to file his answer or other defense in the clerk's office within twenty days after the subpoena has been served on him. Otherwise the bill may be taken *pro confesso*, unless the time be enlarged by a judge, for cause shown. When twenty days is not enough for the preparation of a defense, as in a suit on a patent in a technical art, involving the search for and study of many prior patents and publications, more time will, of course, be given, but the United States judges are all disposed to enforce the spirit of the new rules, and delay cannot be obtained against the opposition of the other side unless some good reason therefor is shown.

The pleadings will be concluded immediately upon the filing of the answer, and the controversy then deemed to be at issue, without the subsequent filing of a replication by the plaintiff, as formerly, unless the defendant sets up a counter claim or set-off in his answer, in which case ten days is allowed the plaintiff to answer this. The case is then ready immediately for the taking of testimony and trial.

As stated above, the testimony is now usually taken orally in open court. That is to say, a day for trial having been set and the parties having prepared their respective cases as fully as possible, in advance, the proceedings will be begun and usually carried through uninterruptedly to a conclusion before the judge.

The plaintiff's lawyer will first address the court, explain briefly his client's patent and outline the case of infringement he expects to prove. The defendant's counsel will then briefly outline the defense he expects to make. Complainant's witnesses will then be sworn, examined and cross-examined by the opposing counsel until the plaintiff's attorney deems he has made out a case. The defendant's witnesses will then be examined, and the defense made clear by the testimony, including, in most cases in which an unadjudicated patent is sued

upon, an attempt to prove the patent in suit invalid in view of certain prior patents or publications which the defense sets forth and explains. The plaintiff then immediately takes his rebuttal testimony, after which the opposing lawyers sum up and the case is left with the court.

This proceeding has taken only a few days, in place of the months and often many years consumed under the old practice in taking testimony in typewriting before a notary public or other so-called examiner, usually in the office of one of the lawyers. The judge has had the opportunity of seeing the witnesses while they give their testimony, of forming his opinions as to their frankness and apparent credibility, and of asking questions when a technical or other point is not clear to him.

He rules immediately upon any question of the admissibility of evidence which arises, and evidence which is clearly immaterial is not allowed to take up time in presentation and space in the record. When he feels that a point has been fully presented, and is clearly understood by him, he will indicate to the examining lawyer that nothing further is required along that line.

At the end of the trial the judge in most cases will have such a clear understanding of the matters at issue that he will be able to hand down his decision from the bench; or if he wishes to take the matter under advisement and hear further from the lawyers by written briefs as to any particular points of law which may have arisen, a shorter time additional than formerly will be required.

The saving of time in this respect will, of course, depend upon the judges themselves and the number of cases they have to hear. If a judge postpones the preparation of his opinion until after numerous other cases have been heard, the saving in this respect will not, of course, be so great. There is now, however, clearly the possibility of getting an opinion from the bench or shortly after the end of the trial, whereas formerly there could not be an opinion until after the judge had considered the records of testimony, etc., which was always done some time after the final hearing.

This form of procedure will doubtless reduce the cost of taking testimony very materially. The records of patent suits under the old practice were often "padded" with immaterial matter and almost always were longer than necessary. This was for the reason, first, that the examiner before whom the testimony was taken had not authority to pass upon the admissibility of evidence. Everything offered was taken down, together with the objections of the opposing counsel, if any.

The lack of power of the examiner was often taken advantage of, and the rules of evidence sadly mistreated by lawyers who wished to get certain testimony upon the record, regardless of its propriety in the case. This greatly increased the cost of the record of the testimony, exhibits, etc., which was usually printed before trial, and also the labor of the judge in reading through the same after trial.

Another reason for the longer records under the former practice was the difficulty of the lawyer in knowing when he had proved enough to make the matter comprehensible to the court, which would later derive its information by reading the record of the testimony. Another cause was the tendency of certain experts and other witnesses to spar and answer by lengthy indirection to embarrassing questions put. These exercises of mental gymnastics have sometimes consumed an excessive amount of time and will certainly not be attempted, or permitted if attempted, when the testimony is taken before a judge in court.

The causes above referred to have often resulted in patent cases in records of many printed volumes of testimony and in lengthy printed briefs submitted by counsel referring to or sometimes quoting at length from the most pertinent parts of the testimony. In such a case it is almost a certainty that the hard-worked judge will not fully read the entire record of the case, but will rely largely on the extracts brought to his attention by the opposing counsel.

Thus, under the old practice, we had the following elements in the trial of a patent case: volumes of testimony taken with

great toil and at considerable expense, with none of which the judge was familiar before the hearing; then the arguments of counsel before the judge, largely directed to giving him the fundamental information required to pass upon the questions involved; then a long period of waiting, in which the judge grappled with the printed records and briefs in whatever time he could find from his study of other records and the hearing of arguments in other cases. At the end of this period the decision was handed down. In one case in my experience the decision was not rendered for about fourteen months after the argument, so that the points made at the argument could hardly have stood out very clearly in the judicial mind at the time the opinion was written.

The new rules cannot fail to make a procedure immensely more rational and expeditious than the above. In one suit with which I am familiar, tried, by consent of the parties in open court in New York the latter part of January, the bill of complaint had been filed the previous October, just three months previously. The suit was on a mechanical device of some complexity, and the theory of operation of this device and those of the prior art set up by the defendant involved the consideration of various mechanical principles. The devices were explained fully to the court by working models, and the whole trial consumed only two days and a half. Decision was reserved in this case, but it is expected very shortly at this writing.

Depositions may be taken out of court under the new practice in exceptional cases only. They will only be permitted upon the application of the party desiring so to take the testimony in cases in which this mode is permitted by statute or when the moving party shows, by affidavit, a good and exceptional cause for departing from the general rule.

Certain sections of the Revised Statutes of the United States provided for the taking of testimony before an examiner, if desired, in the case of witnesses who live more than 100 miles away from the place of trial, and the Supreme Court rules cannot, of course, abrogate this statute. The rules provide, however, that if proper notice of the taking of such testimony has not been given the opposing party, the latter may be entitled to have the witness examined orally in open court if the court deems it proper under the circumstances.

If depositions are taken out of court, under a statute or order of the court, the names of all the witnesses must be given to the court in advance and the testimony must all be taken in a limited time, unless otherwise ordered by the court for good cause shown, sixty days for the plaintiff, thirty more for the defendant and twenty for rebuttal depositions.

The equity rules formerly in force also provided for limited time for the taking of testimony, but this was more honored in the breach than in the observance. As stated, the federal judges now are all determined to strictly enforce the provisions of the new rules with a view to expediting proceedings in all cases in which some good reason for delay is not made to appear.

Special provisions are made for the taking of the testimony of expert witnesses in patent and trade-mark cases. Such witnesses may, and in many cases will, be examined in open court according to the general rule. The court may, however, upon petition order that their direct testimony be set forth in affidavits and filed in a limited time; forty days for the plaintiff, twenty days more for the defendant, and fifteen days more for rebutting affidavits. Should the opposing party wish to cross-examine any of these experts the court may order that such cross-examination and any re-examination take place before the court upon the trial, and unless the expert is produced for such examination his affidavit cannot be used.

Cases are now placed on the trial calendar as soon as the time has elapsed for taking and filing depositions, where these are permitted by the new rules. Thereafter no further testimony shall be taken by deposition except for some strong reason shown by affidavit. That is, even when some of the testimony is permitted to be taken out of court by deposition, as where witnesses live in a distant place, the case will go on the trial calendar after 110 days have elapsed since the case

was at issue. The federal courts all over the country are now calling these trial calendars, and patent suits which have been dragging for a long time are being disposed of or dismissed for want of prosecution.

The Rules of the District Court for the Southern District of New York

The District Court for the Southern District of New York, in which Manhattan Island is situated, have also promulgated new rules, which went into effect on Feb. 1, 1913. It is there provided that in patent cases each side shall be allowed only one expert witness unless leave to employ more be granted on cause shown.

This court also rules that expert witnesses, where their testimony is taken by affidavit as provided for in the Supreme Court rule referred to above, shall confine their testimony strictly to an explanation of the operation of relevant arts, processes, machines, manufacturers, or compositions of matter, and of the meaning of terms of art or science and of diagrams or formulas. They shall not give their opinion as to the meaning of any patent claim or specification, and if matter forbidden by this rule, or irrelevant or immaterial matter is contained in an affidavit, it shall not be answered by the opposite party nor be the basis of any cross-examination and the court may at any time strike such matter from the deposition.

This provision will doubtless result in much shorter expert depositions than most of these formerly taken. The rule is so novel that its actual working out in practice will have to be awaited before its relative merits and disadvantages can be accurately determined.

Another interesting paragraph in the New York rules is one which provides for the engagement of an expert to assist and sit with the judge at the hearing of the evidence in the trial of a patent cause which in the opinion of the court involves intricate technical or scientific questions of fact. In such a case when all parties consent the court will appoint some disinterested person skilled in the art to act as an assessor, as it is termed by the rules, the reasonable fees for whose services are to be fixed by the court and charged as part of the taxable disbursements. The assessor will sit with the judge, assist the latter in his deliberations when requested and render the judge a written opinion when the court desires it, which shall be a portion of the record, on appeal.

It is believed that this rule should prove beneficial, provided a disinterested expert can be found acceptable to both sides in a given case. This may often be difficult, and many parties will prefer to await the trying out of the scheme before consenting to try it in their own cases.

The rule is designed, however, to meet the objection often urged by critics of our patent system that a judge, learned in the law, but not especially trained in technical sciences, should not be left unaided to decide important patent suits, the determination of which may depend upon a nice appreciation of intricate chemical or physical phenomena or a highly technical art.

It is a testimonial to the value of the legal training that judges lacking in special scientific education are as able as they are to lay hold of the important point of such a controversy and rightly determine the merits of the case.

Such an assessor, if broad minded and unbiased, will doubtless be of great assistance to the court in cases in which the opposing experts hold widely differing views. This official should not, of course, usurp the functions either of the court or of the witnesses, and will doubtless merely advise the court, upon request, in the interpretation to be given the testimony and the correlation of technical facts. The working of this innovation will be watched with great interest.

A considerable saving in expense to patent litigants under the new practice should appear in the matter of making up the record. The testimony taken in open court or before the trial need not be printed, as it formerly was in most cases, but will usually be taken down in shorthand by an appointed stenographer, and, if required, transcribed. When a case is appealed to the higher court the evidence to be included in the record

for the upper court will not be set forth in full and printed, as formerly, but will be stated in simple and condensed form, usually in the narrative manner; and those parts only which are essential to the decision of the questions presented by the appeal will be presented. The statement and condensation of the evidence for appeal will be made by the parties under the direction of the court, and all formal and immaterial parts of documents are to be excluded.

The system above briefly outlined is believed to be fundamentally sound and a conclusive answer to many of the criticisms of the former practice. Its advantages will appear in proportion as it is enforced by the courts and lived up to by the parties themselves. Unnecessary delays in patent litigation will occur chiefly when they are agreed to by both sides. Even in such cases suits will not be allowed to slumber in the peace of former years. The District Court sitting in New York City, for example, ordered some time ago that all suits in equity filed since 1902 be brought forth and placed on a special calendar, which was heard the first week in March. This resulted in hundreds of suits which had not been actually prosecuted for several years being dismissed. The same sort of a cleaning-up process is being carried out in the other District Courts of the country.

As to the possible disadvantages of the new system, it may be remarked that cases will have to be prepared in advance with the greatest thoroughness, and the defense anticipated by the plaintiff sufficiently to have the rebuttal witnesses primed and in attendance, since there will usually be no period of delay between the taking of defendant's proofs and the presentation of the rebuttal.

The hardship, however, if any, will be more than counterbalanced by the advantages, among which may usually be included the lack of opportunity to prepare defenses too finely drawn to the facts and opinions educed by the opposing party. The cause of truth will probably be advanced when well-trained witnesses are given no considerable time to revise their views and testimony.

The fact that testimony in open court is not immediately typewritten, as in the usual former practice, but is buried for the day at least in the notebook of the court stenographer will also be embarrassing to some forgetful or uncertain witnesses, but will inure to the advantage of frankness and consistency.

The amount of gain in time required to secure a decision under the new practice will depend somewhat upon the calls upon the time of the judges. The actual hearing of patent cases, will, of course, consume more time now than formerly when in the usual equity case the judge only heard the arguments of counsel upon the evidence previously taken. On the other hand, the judge will be saved a good deal of time previously needed in reading the heavy printed records of testimony. In order to get the best results from the new system it may be that a number of additional federal judges will need to be appointed.

It seems clear that the Supreme Court has accomplished a constructive and well-considered reform by its new rules, for which the inventors and manufacturers of the country should be thankful. It is the more welcome since Congress has so merged and confused the inventor's demand for patent reforms with the supposed popular cry for greater regulation of all monopolies as to result in bills being introduced which would leave the inventor with less rights and remedies than before. There is probably little cause to fear that these bills will pass without material changes, because of the widespread protest against them on the part of patentees and technical interests, but even if they should pass a positive gain has undoubtedly resulted from the action of the Supreme Court.

New York City.

The gold output of the Rand to date amounts to a grand total of about \$1,685,000,000. Of this sum about \$44,375,000 has been paid in dividends. The production in 1912 broke all records, amounting to about \$180,000,000, and equalled about 40 per cent of the gold production of the world.

Iron Pipe Used for Electric Conduit

By Oliver W. Storey

The manufacture of rigid metallic conduit has become an industry of importance to the chemical engineer during the past decade. The competition among the manufacturers of this class of material has become very keen and has helped to develop a variety of methods of treating pipe to make it conform to the requirements of the National Board of Fire Underwriters. As the competition increased, the quality of the finished product increased and now the conduit is of a very high grade.

An electric conduit system may be defined as "a raceway in a building to provide for the ready introduction and removal of electric wires and to provide efficient mechanical protection to them."

This definition requires the following necessary features for such a conduit. It should occupy the least possible space; it should be rugged and mechanically strong, yet be easy to work and install; it should resist corrosive action and it should efficiently protect and preserve the insulation of wires contained in it; and it must be moderate in cost.

Materials possessing all of the qualities mentioned are available but their cost is prohibitive. Ordinary iron or steel pipe is cheap and possesses all of the necessary qualities except resistance to corrosive action, but by treating it to certain processes it becomes available for conduit purposes. It is the purpose of this article to describe these various methods of treatment and their comparative durability.

In the past many substances were used for the manufacture of conduit such as fibre, paper, rubber and moulded pulp. Metal pipe was also lined with paper, wood or compositions but these methods have been practically abandoned for the present practice.

At the present time butt-welded mild steel pipe 10 ft. long is used for conduit. This makes a cheap and excellent base since it is very rugged and can be easily worked.

The present manufacture of unlined conduit may be divided into two general classes, enameled and galvanized. The enameling affords protection to the pipe by a corrosion resisting film of enamel while the galvanizing affords protection by galvanic action.

It is only within the past few years that the sale of galvanized or so-called "white pipe" in contrast to the black enameled or "black pipe" has reached large proportions with a consequent decrease of production of the latter. This condition is caused primarily by the reduction of price of galvanized conduit with increasing competition and also by the general conservation tendency in the iron and steel trade to secure a maximum of protection even with greatly increased cost.

In this article the cleaning of the pipe previous to galvanizing or enameling will not be described. The subject of pickling of conduit pipe is an extensive one and has been discussed in another article.¹ It is assumed that all pipe is clean and in a condition to turn out a good grade of conduit.

Manufacture of Black Enameled or "Black Pipe"

The manufacture of "black pipe" appears as a simple proposition, but the large number of difficulties that appear in carrying out the process of enameling requires a great deal of technical skill to insure success.

The selection or compounding of a suitable enamel is of primary importance. Such an enamel when baked must conform to the specifications of the National Board of Fire Underwriters which are as follows: Section 58, Rule L: "The pipe from which the conduit is made must be thoroughly cleaned to remove all scale and must then be protected against effects of oxidation, by baked enamel, zinc, or other approved coating which will not soften at ordinary temperatures, and of sufficient weight and toughness to successively withstand rough usage likely to be received during shipment and installation; and of sufficient elasticity to prevent flaking when 1/2-in. conduit is bent in a curve the inner edge of which has a radius of

¹Met. & Chem. Eng., Vol. II, No. 1, p. 45.

3½ in. All conduit must have an interior coating of a character and appearance which will readily distinguish it from ordinary commercial pipe commonly used for other than electrical purposes."

Other large users of conduit issue specifications that are usually more rigid. Typical of these are the "Specifications for Conduit and Conduit Fittings" issued by the U. S. Navy Department of May 20, 1912. The sections relating to enameled conduit follow:

- a. The enamel shall be of an approved type and be not less than 0.005 in. in thickness.
- b. Not to be affected by moisture or at a temperature of 100 deg. C.
- c. To be of sufficient elasticity to prevent flaking and to adhere with smooth surface when conduit is bent at normal temperatures in a curve, the inner surface of which has a radius of seven times the normal I. P. S. for ½ in. and 10 times the normal I. P. S. for all other sizes.
- d. Not to be affected by the following after an immersion of 24 hours:
 1. A 30 per cent solution of sulphuric acid.
 2. A 30 per cent solution of nitric acid.
 3. A 30 per cent solution of hydrochloric acid.

The specifications of the Board of Underwriters will allow much cheaper enamels to be used than those of the Navy Department. The latter's requirements are very exacting and only the best of enamels may be used.

The baking enamels used contain linseed oil as a base. The other constituents are tars, asphaltums, pitches, and gums, giving it its black color. The usual thinner is benzine. The enamels are made in many grades depending upon the amount of linseed used. By using a small percentage of linseed oil and a large proportion of asphaltums dissolved in benzine a very cheap enamel baking at a low temperature may be made. By using a high percentage of linseed oil with dissolved gums and asphaltum added an excellent one baking at high temperatures may be made. The addition of small percentages of other materials is practiced to improve certain qualities.

A method of testing for a good enamel is as follows:

Remove a strip of enamel from the pipe with a sharp knife. This strip should approach rubber in its elasticity and yet should be hard enough while on the pipe not to be easily scratched by the finger nail. Even after ageing for several years in a warm dry atmosphere this test should still apply.

The general methods used in conduit pipe enameling are alike and differ merely in details. The pipe is hung on racks and dipped into a tank of enamel, and after withdrawing and dripping is run into an oven and baked.

The clean, dry pipe coming from the pickle room are ready for enameling. The couplings on one end found necessary to secure complete separation of the pipes in the pickling vat are again found necessary to hang them upon the enameling racks.

The racks are usually made of cast iron with a double row of notches into which the pipe will slip easily but upon which the shoulders of the coupling will rest. With small sizes of pipe the notches may be deep enough to support two lengths. The racks are supported by a monorail system. The capacity of each is from 50 to 200 ½-in. pipe and requires a different rack for each size of pipe. Since a large percentage of conduit is ½ in. the number needed for the other sizes is small especially for those above 1 in.

After the pipes have been hung upon the racks they are dipped into the enamel which is usually contained in a riveted iron tank about 11 ft. deep, 2 ft. wide and long enough to accommodate the dipping rack. The tanks often contain 1000 gallons of enamel. With an enamel worth 65 cents per gallon the value of the contents is a considerable sum and precautions must be taken to prevent its becoming ignited, to prevent any deleterious foreign matter entering it or to prevent adding a poorer quality of enamel which necessitates testing of all new lots of enamel.

The danger of fire is increased by heating the enamel by steam coils in the bottom of the tank. The heating is neces-

sary to maintain a constant temperature and to thin the enamel. This method is safer, cheaper and gives better results than thinning with benzine. If a volatile solvent is added the enamel continually changes in consistency with corresponding variation in the finished product. Warm enamel also coats the pipe more evenly by eliminating any air bubbles clinging to the pipe surface.

The rack of pipe is lowered into the enamel and allowed to remain until it has reached the same temperature. It is then hoisted out of the tank and the pipe allowed to drip for about a minute after which they are run over dripping pans and allowed to remain until ready for baking. The excess enamel on the dripping pans flows back into the tank.

With the various grades of enamel this preliminary air exposure accomplishes different purposes. With a high grade, high temperature baking enamel this exposure to the air does not cause any hardening of the enamel but merely allows some of the excess enamel to drain. With a medium grade, benzine-thinned enamel the benzine evaporates, leaving a jelly-like coating that hardens in the furnace with medium temperatures. With a cheap enamel containing a larger percentage of benzine, the latter evaporates leaving an enamel that is but slightly tacky and which becomes hard with a slight baking.

After the preliminary air treatment the pipes are run into the baking oven. These vary in their construction and in capacity which is usually rated in terms of ½-in. pipe. An oven having a capacity of 25,000 feet of ½-in. pipe would hold about 4000 ft. of 4-in. pipe.

The design of the baking oven is dependent upon the enamel used. In general they are made of firebrick with ample ventilation for all fumes and gases evolved.

If a low temperature baking enamel is used steam coils may be placed in the bottom of the oven and adequate heat furnished by these to finish the drying of the enamel.

Where higher temperatures up to 700 deg. Fahr. are desired two general methods are in use, the direct and indirect.

The direct method is used where natural gas is available or other gas can be obtained at low cost. The gas is burned directly in the baking oven in a kind of hearth or brick that are heated to a white heat. The object is secure as complete combustion of the gas with as little excess of air as possible. The hot gases of combustion circulate among the pipe and bake the enamel. With small excess of air there is little danger of igniting the fumes evolved. With incomplete combustion soot would be deposited upon the tacky enamel and spoil it.

With the indirect method of heating the hot gases of combustion are led through radiators which heat the air to be used in baking the enamel.

The efficiency of the direct method is very high when compared to the indirect. With the latter there is no danger of spoiling the enamel with the products of combustion and there is no danger of an explosion should the oven door be accidentally opened during the "heat."

When a medium enamel which hardens slowly in the air is used the oven is run at a temperature of 300 deg. to 400 deg. Fahr. When the oven is first warmed a small amount of enamel may drip from the bottom of the pipe due to the thinning of the jelly-like film left after the evaporation of the solvents.

With one that must be baked at a temperature of 600 deg. Fahr. the operation is not as simple. When the oven warms the enamel thins and the excess will drip to the bottom of the furnace. This dripping will continue for some time, usually until the solvents have evaporated and the linseed oil has begun to harden. The excess is drained into a well from which it is periodically returned to the tanks, apparently having suffered no deterioration.

The securing of a uniform film of enamel over the surface of the pipe is a difficult problem. After baking a rack of ½-in. pipe, that have received one coat of enamel, the following imperfections will be noted:

1. The upper end of the pipe, both interior and exterior, has received a thinner and harder coating than the lower end.

2. The threads at the lower end are filled with enamel.
3. The surface of the exterior of the pipe that is very close to another pipe has a coating thin enough to be transparent, while the balance of the pipe has a heavier coating.
4. The interior surface of a piece of $\frac{1}{2}$ -in. pipe has received a thin coat while the same surface on a piece of 4-in. pipe is as heavily coated as the exterior.

These are the chief difficulties encountered in enameling conduit.

The coating at the upper end of the pipe is usually half as heavy as that at the lower end. This is caused by the upper part of the enameling oven becoming hotter first and the draining of the surplus enamel to the bottom of the pipe. The enamel flows slowly to the lower end, and, before it can all drip, it has begun to harden since the rate of heating is usually high. With slower heating this difficulty could be lessened. At the upper end of the pipe it is also harder than at the lower owing to the thinner film and higher temperature.

These imperfections may be remedied in the second application by upending the conduit on the racks. This is not generally done owing to the extra operation of changing couplings which would require that the racks be unloaded and reloaded. Some manufacturers depend upon the second coat being more uniform than the first and covering any pronounced difference in the upper and lower end.

The unevenness of coating due to the proximity of other pipe can obviously only be remedied by cutting down the capacity of the oven. Since the second coat of enamel covers this defect it is usually disregarded. If but one coat of enamel were used this defect would have to be remedied.

The difference of thickness between the interior and exterior coat is primarily caused by the lack of circulation in the interior of the pipe and consequent slow evaporation of the solvents and slow oxidation of the oils, allowing more enamel to drain. The enamel film is also very thin at the top when compared to the bottom. As the size of the pipe increases these difficulties decrease.

The two coats of enamel are necessary to give a flexible coat of suitable thickness as required by the specifications and to overcome some of the defects mentioned. A coat of the requisite thickness could not be applied in one baking for the same reason that a thick coat of paint is never used.

The second coat is applied in the same manner as the first and results in a bright smooth surface. This coat is usually baked at a slightly higher temperature than the first giving a harder surface, yet flexible owing to the softer foundation. A third coat is sometimes required but such an additional film is of doubtful value as an added protection.

The conduit is now ready for the market after an inspection of the bore to discover obstructions. This inspection is carried out by flashing a light through it.

The enamel interior provides a smooth raceway for wires. Its generally excellent appearance, its cleanliness, smooth interior and its cheapness makes the black pipe very popular among the installers.

The racks used for dipping the pipe soon become covered with enamel and unfit for use. This can be easily removed by burning in a scrap fire.

Elbows are enameled by running wires through the bores and dipping them as with pipe.

The enameling plant should be so planned that the process may be carried on without interruption during the different operations. The layout should allow enough space to provide for the hanging of an oven-full of green pipe and enough dripping space to take care of an oven-full of dipped pipe. The mechanism should be so arranged as to allow a batch of baked pipe to be removed from the oven while a second batch is run into it. During the baking of this, the first can be given its second dip and allowed to drain.

Since the quality of the enamel is dependent in a large measure upon the temperature control of the ovens, recording gauges should be used to keep an accurate record of the temperature at various parts.

Methods of Manufacture of Galvanized or "White Pipe"

While the general practice in the manufacture of "black pipe" is essentially the same at all plants, the practice in the "white" or galvanized pipe industry varies widely. This difference is due primarily to the three galvanizing methods used, these being hot galvanized, electro-galvanized and sherardized. Approved conduit of all these three types are on the market, the electro-galvanizing predominating since the one sherardized conduit is patented, and there has been but one **hot galvanized conduit** approved by the underwriters.

The specifications of the Board of Underwriters have been quoted and are easily complied with. In direct contrast are the rigid specifications of the Navy Department which exclude hot galvanizing.

"2. h. All steel conduits shall have the exterior surface galvanized and the interior surface either enameled or galvanized and enameled. All galvanizing shall be done either by electro-plating process or dry galvanizing process, and shall show no signs of rust when conduit is immersed in warm salt water (90 deg. C.) for 48 hours. The outer surface of conduit shall be free from paraffin or other material which would tend to prevent the adhesion of paint. Before any tests are made, conduit shall be washed thoroughly in carbon tetrachloride.

"3. d. 4. When conduit is both enameled and galvanized on inside tests under 3. (d) 1, 2, and 3 (acid tests) will not be applied."

The chief trouble that has been encountered in the manufacture of hot galvanized conduit has been the wiping of the interior of the conduit to secure a thin, flexible, and adherent coating. The present hot galvanized conduit on the market has a smooth wiped interior and exterior surface while the threads are clean, yet well coated and need no recutting. The coating is somewhat heavier than is found on electro-galvanized or sherardized pipe yet it is flexible and adheres well to the iron base.

This coating contains aluminium in varying amounts, often being as high as 10 per cent. The addition of this metal improves the hot galvanizing bath without materially increasing the cost owing to its low specific gravity. The aluminium gives the finished conduit a pleasing white appearance.

The presence of about 4 per cent of aluminium reduces the melting point of zinc from 788 deg. Fahr. to 716 deg. Fahr., and allows the bath to be run at lower temperature and decreasing the amount of zinc-iron alloy formed. Since 4 per cent is the eutectic percentage any other low percentage of aluminium will make the solidification occur over a shorter or longer range of temperature depending upon the composition and will, therefore, make it easier to wipe.

The interior of this hot galvanized conduit is not enameled but a peculiar blue black wash is used. It is a thin coating and is merely used as a distinctive finish and helps the fishing qualities slightly.

The one **sherardized conduit** is made by the sherardizing process and has a coating of zinc-iron alloy on the interior, exterior, and threads. While it is not as heavy as that obtained by the hot galvanizing process it is much heavier than that obtained with the electro-galvanizing. It adheres firmly to the iron base, does not flake and is hard, making it capable of withstanding rough usage.

Analyses of the zinc-iron alloy made at different times will show varying amounts of iron present. The iron varies with the metallic zinc in the zinc dust, thickness of coat, sherardizing temperature and length of time in the furnace. The amount of iron in the alloy has been subject to a great deal of discussion since the different investigators have found widely varying percentages. Patrick and Walker² found 11.70 per cent, Leighou and Calderwood³ 18.15 per cent, while Cushman⁴ found about 30 per cent. In none of these cases was

²J. of J. & E. Ch., Vol. 8, No. 4, p. 239.

³Reports of Tests on Sherardized and Electro-galvanized Conduits, Carnegie Inst. of Tech.

⁴Efficiency of Zinc Coated Conduit Pipes, Inst. of Industrial Research, Washington.

it known under what conditions the alloy was formed. From a large number of analyses made by the author it was found that the above variables determined the iron content of the alloy formed. By controlling these factors any zinc-iron alloy containing from 8 to 30 per cent of iron may be obtained.

Sherardized conduit receives an exterior and interior coat of clear enamel. Previous to this practice a coat of clear enamel was put upon the interior only and the exterior was paraffined. The clear enamel was used to aid fishing while the paraffin was used to lay the objectionable loose zinc dust that remained on the pipe.

Electro-galvanized conduit is made by varying methods. Only the exterior is plated since the plating of the interior has not been put on a commercial basis. Usually the threads have little or no coating on them, the interior of the couplings are rusty and elbows are poorly coated. With one exception the zinc coating is thin.

One brand of conduit is made by flashing the cleaned iron pipe in a copper bath and plating over this copper with zinc.

Another manufacturer obtains a smooth velvety adherent coating by wire brushing the pipe before coating with zinc. A second brushes the conduit after plating and obtains a bright finish which is pleasing to the eye and does not discolor easily.

All "white pipe" is enameled in the interior to help the fishing and with the cold galvanizing to provide against oxidation of the iron. In the Navy Specifications the acid tests do not apply to the enamel in the interior for that conduit which has a protective zinc coating beneath. This allows a very thin enamel to be used by those manufacturers who apply zinc to the interior wall while a good grade giving a heavy film must be used by others. This somewhat equalizes the costs.

The coatings used in the interior of conduit cover a wide range. The colored wash in the hot galvanized and clear enamel in the sherardized have been mentioned. Electro-galvanizers use a black enamel except one who uses a red iron oxide enamel paint which is of an excellent quality.

To enamel or paint the interior of conduit requires a special apparatus. A tank similar to the one used for black pipe is used. The pipes are let down into this on racks. The enamel is stored in a small tank about 10 to 15 ft. above. A circular spraying nozzle is let into the top of the pipe. A valve allows the enamel to flow and the head of enamel sprays it on the interior wall of the conduit. Enough is introduced to thoroughly cover the entire interior surface. The surplus drains to the bottom of the tank from where it is pumped back to the storage tank to be used again. This method of enameling is very effective and one man can work at the rate of 600 lengths of pipe per hour. The remaining procedure is similar to that of the black pipe.

Enameled or painted "white pipe" is now sold extensively. The enameling of this pipe does not present any difficulties beyond those found with the black pipe. The applying of a paint presents difficulties of a different nature. The pigments used settle quickly and some method must be devised to keep it evenly distributed throughout the oil. The stirring may be accomplished by means of a propeller. Even with stirring the conduit will not be evenly coated by the pigments. The more practical way of applying the paint would be by means of a spray. In this way the pigment would be more evenly distributed and not be washed away as it is when the conduit is withdrawn from the tank.

When sherardized pipe was paraffined the following method was used. A solution of paraffine of the requisite strength was made in warm benzene. This solution was contained in a horizontal tank capable of holding 5000 ft. of $\frac{1}{2}$ -in. conduit. Steam coils kept the solution hot to prevent the paraffine from freezing out. About 5000 ft. of $\frac{1}{2}$ -in. pipe, after enameling the interior, were dipped into this tank in one bundle. Any superfluous dust was washed off and settled to the bottom while a thin layer of paraffine remained on the pipe after the benzene had evaporated.

After the conduit has been galvanized and enameled it is inspected, labeled, bundled and is ready for shipment.

Black Pipe versus White Pipe

At present the relative merits of the various brands of "white pipe" are subject to a large amount of controversy. The "black pipe" has been almost entirely neglected, it being generally taken for granted that it is an inferior conduit. Large numbers of tests of galvanized pipe have been made both by partial and impartial investigators and the results made by the former are contradictory while those made by the latter agree fairly well.

If the "man on the job" were asked what pipe he would prefer there would be but one answer which would be "black pipe." He prefers it because of its clean, smooth surface, superior appearance, excellent fishing qualities and because it "takes" paint. He objects to the rough surface of the electro-galvanized because of the "acid" which he says eats into his skin. He objects to the hot galvanized pipe because of its rough surface and some even complain of the "acid." The sherardized paraffine finished pipe was objectionable because of the paraffine finish which became soft in warm weather. Without the paraffine the conduit was dusty.

The stiff competition in conduit manufacture has raised the standard to a high plane. Where, about six or seven years ago, most any kind of enameled pipe would have been acceptable, now this is no longer true and a good quality has to be made. By way of confirmation, one only needs to look at conduit installations of a few years ago and compare them with those of to-day.

As stated above the durability of the different kinds of "white pipe" has been the subject of much argument. The relative merits of black and galvanized pipe are not often mentioned and the writer wishes to emphasize the high merit of the former. Each class of pipe is best under certain conditions and before an installation is made these should be studied to obtain the greatest service per unit of cost.

Conduit is subject to a wide range of conditions. Outdoors it is subject to the action of the weather; it may be buried and here a variety of conditions may affect it, ranging from a dry to a wet, acid soil; it is subject to temperature extremes; it may be buried in cement, cinder, concrete or various plasters; it may be exposed to corrosive vapors, coal smoke, or gases; it may be attacked by oils, acids or alkalis, and it is always subject to mechanical injury in transportation during installation and sometimes in service.

Prof. C. F. Burgess³ has shown that a high grade enameled conduit, the surface of which has not been injured, is superior to galvanized conduit under a variety of corrosive conditions. Those cases coming under the writer's observation have confirmed this, but only where a good grade of enamel was used. Any enamel that conforms to the Navy Specifications as given above must be resistant to corrosive influences.

But under ordinary conditions of shipment and installation the exterior enamel is scratched and scraped, exposing the iron beneath and subjecting it to corrosion. This condition is necessarily important when comparing with galvanized conduit.

Under the following conditions enameled pipe is cheaper to install and will be fully as satisfactory as the galvanized. In room interiors where the conduit is afterward painted, under similar outdoor conditions, in walls of buildings including both plastered and concrete where moisture is not present; in short, any place where dry conditions prevail and any exposed iron is not subject to permanent corrosive conditions. This would cover a large percentage of installations.

By painting the enameled pipe after installation it could be made superior to non-enameled galvanized conduit under many conditions. Where the latter is installed and is exposed to acids and fumes the galvanizing soon is corroded. The resistance of unscarred enameled pipe to such conditions has been pointed out by Prof. C. F. Burgess³. By giving the scarred places a coat of slow air drying enamel after installation, the conduit could be made comparatively permanent.

Under exposure to alkalis there would be an advantage to the galvanized conduit but under such conditions even the latter

³Sherardizing Magazine.

shows a high rate of corrosion and brass, lead, or composition conduit are necessary. Under conditions where the application of this enamel after installation would be impractical or under peculiar conditions of exposure where the enamel would blister and peel the galvanized conduit would be better suited.

The importance of securing as permanent an installation as possible in modern buildings even at a much higher cost has been recognized by some engineers and architects. These men have specified conduit that is both galvanized and enameled, and, so insistent has been this demand, that manufacturers are turning out this class of conduit.

The importance of such a protecting coat over the galvanizing was first noticed in the testing of the paraffined sherardized pipe. The paraffine was primarily used for laying the dust but at the same time it increased the resistance to corrosion to a remarkable degree. It seems as though this increase in the life of the sherardizing would have been the important feature but was not recognized as such by the manufacturers. Tests^{3, 4} on paraffined and non-paraffined conduit showed the marked superiority of the former, being fully as durable as the enameled conduit.

The writer believes that any enamel that is used over the galvanizing should be equal in quality to that used on "black pipe." A cheap enamel will soon disintegrate and leave the galvanizing exposed. And where such a conduit is installed the scarring of the enamel during installing will make it little better than the raw galvanized material. The painting of all scars is very important and should be rigidly enforced if the maximum of service is to be received.

The U. S. Government has recognized the value of such a double protection, especially on war vessels, and along the Panama canal, where galvanized conduit, usually sherardized, is painted after installing and is repainted regularly.

The value of the different galvanized coatings when used for conduit protection is a much discussed subject. With the constantly improving methods and with the new brands continually appearing, no data are ever complete.

Some of the earliest work was done by Prof. C. F. Burgess⁵ at the Chemical Engineering Laboratories of the University of Wisconsin and showed the relative merits of hot, cold and dry galvanized and enameled conduit. Messrs. Leighou and Calderwood⁶ of the Carnegie Technical Schools made some exhaustive tests in 1912 of the value of electro-galvanizing and sherardizing as a protection for conduit purposes.

The value of galvanizing is usually determined primarily by the amount of zinc per unit of area. The other important factors are homogeneity, purity, continuity and tenacity with which it clings to the iron.

The cold galvanized conduit is made by several manufacturers and the product varies. The first of such conduit showed up poorly under actual service conditions as well as in accelerated tests. This was the result of the light deposit of zinc and its uneven distribution. The threads and couplings were usually subject to immediate corrosion owing to the absence of zinc. The amount of zinc has been materially increased but even now the threads and couplings rust quickly. The writer has under his direct observation a large amount of cold galvanized conduit of recent installation and every length is badly rusted at the joints.

The copper flashed conduit employing a thin zinc deposit has shown no superiority over the ordinary electro-galvanizing under actual service conditions.

The sherardized conduit has been subject to many tests and has shown that when the alloy is formed *under proper conditions* it results in a coating that is superior to the electro-galvanized or hot galvanized material. The present practice of sherardizing shows a lack of uniformity of practice and results obtained. The theoretical alloy that should be obtained is one corresponding to FeZn_{10} , containing about 8 per cent of iron. Where this alloy has been obtained a coating highly resistant to corrosive action has resulted. The writer's experience has shown that a low temperature is necessary for the production of such an alloy, probably lower than any temperature

used in commercial sherardizing plants of today with but one or two exceptions.

With the increase of iron content of the coating the more quickly will it corrode. With coatings containing from 25 to 30 per cent of iron the writer has found that the sherardizing affords little protection since it pits easily. Such a coating is obtained by using high temperatures and a lean zinc dust.

A peculiar property of this alloy is the formation of a blue black covering after being exposed to the weather for some time. Cushman⁷ calls attention to the highly protective powers of sherardizing, through its ability to take on this highly resistant black coating, which he explains as being the formation of Fe_3O_4 from the oxidation of the iron in the alloy. He states that this oxide will greatly increase the life of the coating owing to its superior weathering qualities. This black appearance is especially found on exposed paraffined conduit.

The successful introduction of the hot galvanized conduit is of great importance since it is almost impossible to obtain a thin coating with this process. This conduit should, if the weight of coating and other factors are considered, have a long life. Hot galvanized pipe stands high in corrosion tests.⁸ What effect the introduction of aluminium will have can only be conjectured.

The importance of the interior galvanizing is often over-estimated. In most cases few fumes and vapors come in contact with the interior and a coat of good enamel should be sufficient to protect against all corrosive influences.

The testing of galvanized conduit is usually done by the standard Preece copper sulphate method and the coats are usually spoken of as being so many "dips" in thickness.

The writer has tested the Preece method and has found that it is very reliable, cheap, and entirely satisfactory as a quick factory method for testing galvanized conduit. The heaviness of the coating is a straight line function of the number of dips and makes the factory control very simple.

But results obtained by this method on two different types of galvanizing are not comparable. It will be found that sherardizing of the same weight as electro-galvanizing will give twice the number of dips in copper sulphate.

The writer has often heard the statement that a coating standing one dip will be equal to three years of atmospheric corrosion. On the strength of the copper sulphate tests resulting from the above statement, a large amount of conduit has been sold. Some manufacturers, quickly realizing that the addition of certain materials to the coating would raise the Preece test without materially increasing the coat, began this practice. As a result of this the test when used by the salesman is practically worthless.

To arrive at a fair basis, depending upon the actual amount of zinc present, a new method has been proposed.⁹ In this method the zinc or zinc iron alloy is removed by basic lead acetate. This method is very satisfactory where an accurate determination is desired and in the comparison of two unlike conduits. For routine factory testing it is not practicable since it involves several weighings with the balance. It may be used in occasionally checking up the copper sulphate tests. The actual amount of zinc present will not be indicative of the serviceability of the conduit for this will depend on the process.

From this description of the making of rigid conduit after leaving the tube mill the process is seen to be a product of the chemical engineer and electrochemist. Unfortunately they have not been instrumental in the early development of the processes that have only received their aid during the past decade.

How long the galvanized conduit will be supreme is a question of when some other cheap resistant coating may be applied to iron or some other resistant metal or alloy may be produced cheaply enough to compete with steel. The possibility of using a lead coating instead of zinc is an attractive proposition. Samples of such conduit have come to the notice of the writer from time to time and to all appearances should make an excellent conduit.

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The Determination of Vanadium in Ferro-Vanadium

By Wm. W. Clark

As all ferro-vanadium is sold on a basis of the vanadium content, the determination of vanadium is the most important one in the analysis of the alloy. It is not a difficult determination, but is one that is easily influenced by conditions, and consequently very few analysts get correct results by a few tenths of a per cent.

A number of methods have been introduced for the determination, which when run on pure salt, give fairly concordant results, but when an alloy of indefinite composition is encountered, the chances of error multiply, or if the vanadium is separated from the other metals present, considerable time is wasted, with corresponding losses of vanadium in the separation. The majority of the methods depend upon the oxidation of the vanadium from the vanadyl to the vanadic salt or vice versa. Some of these methods are indirect, such as the reduction of the vanadic salt to the vanadyl, with hydrochloric, hydroiodic, or hydrobromic acid and the absorption of the evolved chlorine, iodine or bromine in potassium iodide, and subsequent titration of the liberated, or absorbed iodine with sodium thiosulphate solution. These methods are erratic on account of the reduction not stopping at a specified compound, some of the vanadium going to a lower valency, with the liberation of the corresponding amount of the halogen.

The method which seems to be in general use is the separation of the vanadium from the metals by means of sodium hydrate, reduction of the acidified filtrate with sulphur dioxide, removal of the excess sulphur dioxide by boiling and a current of carbon dioxide, and subsequent oxidation of the reduced vanadium with potassium permanganate. This method, while theoretically correct, gives results which as a rule are high, if the vanadium is all separated from the precipitate of iron. It requires two or more separations to get all the vanadium out of the bulky iron precipitate, and the filtrate must be boiled and refiltered on a very close filter as some iron always goes through on the first filtration. The method is slow, taking not less than four hours for a single determination, and most all of the operator's time. It takes several filtrations, which we try to avoid as much as possible, as they take time and introduce errors. The method in detail is as follows:

A factor weight of the finely powdered alloy is dissolved in sulphuric and nitric acids, evaporated to white fumes of sulphuric anhydride, diluted, and the iron precipitated by pouring the nearly neutralized solution into an excess of boiling sodium hydrate, which contains a small amount of sodium peroxide. Sodium should always be used when separating vanadium, as sodium vanadate is more soluble than the potassium salt. The solution is boiled and filtered. The precipitate is dissolved in hot sulphuric acid and reprecipitated. The combined filtrates boiled, again filtered on a close filter, acidified with sulphuric acid, adding about 10 per cent excess, reduced with purified sulphur dioxide, the excess of which is removed by boiling and the passage of a current of carbon dioxide, which has been passed through a solution of sodium carbonate. The gas is passed until the steam will not discolor a solution made slightly pink with a drop of permanganate. The vanadium is then oxidized with N/10 potassium permanganate, titrating to a pink in the hot solution.

Following are typical results on three different vanadium compounds: a pure solution of vanadium, a synthetic solution made of approximately the same composition as a solution of an alloy, and a standard ferro-vanadium. The pure solution was made from pure vanadium pentoxide, which was made by recrystallizing ammonium vanadate several times. The salt was dried and the major portion of the ammonia driven off in a drying oven. The oxide was then melted in a platinum dish in a current of oxygen. The resulting oxide was ground to a fine powder and showed the following composition by analysis:

Vanadium	55.96%
Vanadium Pentoxide	99.86

Vanadium Tetroxide	0.00
Ammonia	0.00
Sodium Oxide	0.048
Silica	0.063
Alumina	0.041
Alkaline earths	0.00
Hydrogen sulphide metals.....	0.00
Iron group	0.009

A weighed amount of this oxide was fused with pure sodium carbonate, the water solution acidified with sulphuric acid, and diluted to a definite volume, 1 c.c. of which contained .003136 grams vanadium. The synthetic solution was made from this solution, and C. P. salts. The standard ferro-vanadium has been carefully analyzed and the results averaged. Fifty c.c. of the pure solution was taken in each case, this being equal to a percentage of 30.74 vanadium on a factor weight of alloy.

Pure solution Vanadium present.	Synthetic solution Vanadium present.	Standard Ferro-Vanadium Vanadium present.
30.74%	30.74%	35.56%
Vanadium found.	Vanadium found.	Vanadium found.
30.68%	30.82%	35.74%
30.74	30.92	35.86
30.78	30.88	35.74
30.70	30.82	35.88
30.78	30.96	35.72
Average results	Average results	Average results
30.74	30.88	35.79

The pure solution gives very good results, but the synthetic solution and the standard ferro-vanadium show an increase of 0.14 per cent and 0.24 per cent respectively. This increase of vanadium has always been found, and if the work is very carefully done, keeping all conditions constant, the results will keep down between 0.10 per cent and 0.30 per cent high, but at the present price of ferro-vanadium this is too great an error. It has been noticed that often when no more sulphur dioxide can be detected in any way, the results will run several per cent high. This has not been accounted for, except on the theory that some product is formed in the solution which is oxidized by permanganate, nor has the cause of the ordinary high results been determined. Of course this method cannot be used when metals reducible by sulphur dioxide are present, as they will again be oxidized by the permanganate with the vanadium.

All the other methods, which were at all practical, were tried in the same manner, and discarded for very much the same reasons as the above method.

The following method has been found to be accurate under all conditions, and on all varieties of work. The solutions needed are:

N/10 potassium bichromate, made by dissolving 4.908 grams of the pure salt per liter of water, and carefully standardizing to exactly N/10 at 20 deg. C. against pure iron wire, and also against N/10 potassium permanganate which has been standardized at 20 deg. C. against sodium oxalate, procured from the Bureau of Standards. Ferrous ammonium sulphate was used as an intermediate solution. The bichromate is constant and needs very little care after it is standardized, except that it should be restandardized once a week to guard against accidents.

N/10 ferrous ammonium sulphate solution, made by dissolving 40 grams of the salt per liter of 10 per cent sulphuric acid. As the strength of this solution changes about .004 c.c. per c.c. a day, it is standardized against the bichromate under working conditions every day.

A blank of 0.32 grams pure iron is run the same as a test to the point of titration, when 35 c.c. of the ferrous solution is added. This is titrated back with the bichromate to a definite end point. For instance 35 c.c. of the ferrous solution took 34.20 c.c. of the bichromate to oxidize it. This shows the ferrous solution to be 0.80 c.c. weak in 35 c.c., or 0.023 c.c. per c.c.

This factor is subtracted for each c.c. used on a determina-

tion, or if the ferrous solution is strong, the factor is added.

Potassium-ferri-cyanide solution made by taking a piece of the salt about the size of a pea, and washing the outside several times with pure distilled water to remove any ferri-cyanide which may have been formed on the outside, and dissolving the remainder in 50 c.c. distilled water. The ferri-cyanide must contain no ferro-cyanide, as vanadyl salts gives a green precipitate with ferro-cyanide and this can easily be mistaken for the green of the ferrous iron. A paper appeared some time ago (Cain, J., *Ind. & Eng. Chem.*, Vol. III, 476), claiming that vanadyl salts in acid solution, reduced ferri-cyanide to ferro-cyanide, thus giving erratic results. This has been tried from every angle on pure salts and under working conditions, and no immediate action could be noticed. Fifty c.c. of a 10 per cent solution of vanadyl sulphate was added to 50 c.c. of a 1 per cent solution of ferri-cyanide, and allowed to stand for twenty-four hours. No precipitate had formed at the end of that time, while when one drop of a 1 per cent solution of ferri-cyanide was added to the mixture, a light green precipitate developed in a few minutes. The presence and amount of ferro-cyanide in ferri-cyanide can very easily be determined by means of this reaction. Care must be taken not to make the ferri-cyanide solution too strong as the yellow color will mask the last traces of green in the spot.

Potassium permanganate solution, made approximately N/10 but not standardized.

Sulphuric acid—one part acid to two parts water. This strength acid is used, as it can be poured into hot fuming sulphuric acid without spattering, saving considerable time cooling and again heating the solution.

The apparatus used is an automatic burette, for the ferrous ammonium sulphate solution, made from our specifications by Bausch and Lomb. The burette is made to deliver 50 c.c. and is graduated in 1/20 c.c. from 30 c.c. to 50 c.c. A 25 c.c. burette graduated to 1/20 c.c. is used for the bichromate and an ordinary 50 c.c. burette for the permanganate. A white porcelain spot plate can be used. The depressions should be shallow as it is easier to control the spot in a shallow one. They are cleaned by dropping into a sodium hydrate solution after using, and then washing. The best spot is shown on a piece of waxed dead white, high gloss cardboard. The card is cut to size and immersed in a solution of white paraffine, repeating until the coat of wax is heavy enough to keep the solution from coming in contact with the paper.

The drop remains spherical and reflects the smallest traces of green. The waxed cards are easier to handle than the plates, and after being used are thrown away. The ferri-cyanide is kept in a flat top dropping bottle, which regulates the size of the drop.

The method is as follows:

Dissolve a factor weight (0.510 grams) of the finely powdered alloy in 25 c.c. of the dilute sulphuric acid and 10 c.c. concentrated nitric acid, and evaporate to white fumes of sulphuric anhydride. If the alloy is refractory to these acids, it is transferred to a platinum beaker with the addition of 20 c.c. hydrofluoric acid, and again evaporated to white fumes. Seventy-five c.c. of the dilute sulphuric acid is added to the hot fuming mass and the solution heated until all the salts have gone into solution, when it is boiled for two minutes. The solution is transferred to a 5-in. white porcelain casserole, 75 c.c. more of the sulphuric acid added, and hot water until the bulk of the solution is from 400 to 500 c.c. The solution is then heated to above 65 deg. C. Potassium permanganate is added from the burette to a deep red. This red is just discharged with the ferrous solution, then permanganate is added drop by drop until the last drop just gives a pinkish tinge to the solution, which is discharged by one drop of the ferrous solution. The ferrous sulphate burette is now filled to O, and an excess is run into the solution, this excess is shown by the ferri-cyanide spot. An excess of at least 2 c.c. should always be added. This excess is titrated back with the bichromate solution being careful to come to the same end point as in the blank. The c.c. of the ferrous sulphate solution

used, plus or minus the correction, less the c.c. of bichromate solution used to oxidize the excess of the ferrous solution, gives the percentage of the vanadium in the alloy directly.

The only points to be carefully watched are the neutralization with the ferrous and permanganate solutions, the end point with the ferri-cyanide spot, the variation of the strength of the ferrous ammonium sulphate solution, and the temperature of the solution while being titrated.

The neutralization point can be caught within 1/20 c.c. after a little practice. The operator can alternate drop for drop of the two solutions as long as he is uncertain as to the exact point, without impairing the accuracy of the determination in the least. An alternate method can be used for this neutralization as follows: The alloy is dissolved in 25 c.c. of the sulphuric and 10 c.c. nitric acid, evaporated to white fumes, the salts dissolved in hot water, and diluted to 500 c.c. Potassium permanganate is added in excess, and the solution boiled until all permanganate is decomposed, or a crystal of manganous sulphate can be added to hasten the decomposition. The precipitated manganic oxide is filtered on a tight asbestos plug, using suction. The filtrate is boiled, and if any precipitate comes down, it is again filtered. The solution is now evaporated to a convenient bulk (about 300 c.c.), 150 c.c. of the dilute sulphuric acid added, the solution heated to above 65 deg. C., and an excess of ferrous solution added directly. This method gives good results, provided all the manganic oxide is removed by filtration, but takes at least an hour longer than the regular method.

The end point with the ferri-cyanide is very easily seen, and if the operator bears in mind the color of the end point on the blank, and on the standardization, and always titrates to that point he will get good results. No two operators see the end point the same, so they must use their own standardization, and blank. If the end point is over titrated more ferrous solution can be added and again titrated back with bichromate.

The temperature of the solution must be kept above 65 deg. C. or the results will be low. Tests run at different temperatures show that the amount of ferrous solution oxidized by the vanadium decreases with the temperature, running as low as .60 c.c. in the cold, but it makes no apparent difference how far the temperature is raised above 65 deg. C.

The above method is fast, taking about 30 minutes for a single determination, does not vary when other metals are present, chromium being the only interfering element, and its interference can be eliminated by getting the neutralization point between the ferrous and permanganate solutions in the cold, and then heating to above 65 deg. C. for the titration. The presence of chromium can be shown with an acetic acid solution of diphenylcarbazine, this reagent being very sensitive to chromium.

Each and every determination of vanadium made on ferro-vanadium is run in duplicate, and not once in a hundred times is there a difference of more than 0.10 per cent.

The following results obtained on regular routine work, show how the method runs.

Pure solution	Synthetic solution	Standard
Vanadium present.	Vanadium present.	Ferro-Vanadium
30.74%	30.74%	Vanadium present.
Vanadium found.	Vanadium found.	Vanadium found.
30.76%	30.72%	35.56%
30.70	30.78	35.58%
30.74	30.80	35.50
30.75	30.72	35.60
30.78	30.74	35.54
30.79	30.73	35.58
30.72	30.76	35.60
30.72	30.80	35.58
30.76	30.80	35.54
30.75	30.72	35.52
		35.55

The following results were taken from one of the chemist's record books, and represents one day's routine work on the

determination of vanadium in ferro-vanadium, showing how close it is possible to duplicate.

Determination.	Duplicate.
36.70% V	36.70% V
30.44	30.44
39.47	39.48
38.35	38.36
28.22	28.24
34.13	34.13
38.24	38.26
30.53	30.51
29.78	29.83
28.41	28.43
37.22	37.24
26.69	26.80
36.79	36.81
29.33	29.36
37.06	37.01

American Vanadium Company,
Pittsburgh, Pa.

Spelter Production in 1912

The Geological Survey has just issued its advance statement of spelter production and consumption in 1912. This statement gives the final figures of output by the zinc smelters for the year, distributed both by states producing the ore and by states in which the ore was smelted, thus giving a measure of the zinc mining industry as well as the zinc smelting industry. The zinc ore made into pigments is not included in this statement, hence the full extent of the zinc mining industry is not covered. The imports and exports of spelter, zinc dross and zinc ore are also given, as well as a list of smelters and their capacity, revised to the close of 1912, together with additions being built during the first months of 1913. A long chart shows graphically the fluctuations for the last seven years in the price of spelter at St. Louis and London, and in the price of 60 per cent zinc concentrates at Joplin.

The production in the United States of spelter made from ore, both domestic and foreign, was 338,806 short tons, an increase of 52,280 tons, or 18.2 per cent, over that of the previous year, and by far the largest output in the history of the industry. The final figures show that the Survey's estimate of production given out January 2, 1913, was too low by only 176 tons, or 0.05 per cent. The spelter made in the United States from foreign ore in 1912 amounted to 14,899 tons, almost exactly the same as in 1911, and considerably less than in 1910 and several preceding years. The production of spelter from secondary sources such as skimmings and drosses also made large gains, being estimated at 50,000 tons, of which 21,000 tons was redistilled, partly at regular zinc smelters using ore and partly at plants devoted exclusively to the redistillation of secondary metals.

The consumption in the United States of spelter made from ore was 340,372 tons, an increase of 60,313 tons, or 21.5 per cent, over that of the previous year. The increase in consumption was made possible by the large imports of spelter. For the last four months of the year the average St. Louis price of spelter was more than 1½ cents above the London price. It was during this period that the larger part of the 11,115 tons of foreign spelter was imported.

The list of smelters shows a total capacity of 107,948 retorts at the close of 1912, with additions of 12,216 retorts under construction. With the exception of 576 retorts, the additions are all in Illinois. It should be borne in mind that all this capacity will not be effective for smelting ore, for several of the plants listed are devoted partly or exclusively to the recovery of spelter from secondary materials.

An aluminium solder has been patented by Richard Seifert, of Union Hill, N. J., consisting of 38 per cent pure tin, 32 per cent phosphor-tin and 30 per cent pure zinc. It is claimed that this solder will be effective in uniting pieces of aluminium, or pieces of brass or bronze with aluminium.

Electrolytic Production of Iron Sheets and Tubes, Etc.

By Wilhelm Palmær and J. A. Brinell

Upon request of the directors of Jernkontoret (the Swedish Iron Masters' Association) the authors have made a study of the S. Cowper-Coles electrolytic method for producing iron sheets and tubes, etc., at the inventor's laboratories in London. The results are given in the following paper. For the purpose of investigating the quality of the products, the composition of the electrolyte and other matters, tests have been made at the testing laboratories of the Royal Technological Institute at Stockholm, the results of which tests also will be given in the following.

The inventor has described his method in a lecture: "The Production of Finished Iron Sheets and Tubes in One Operation," delivered at a meeting of the Iron and Steel Institute at Middlesbrough in 1908, published in the *Journal of the Iron and Steel Institute* and in *Electrochemical and Metallurgical Industry*, November, 1908, vol. VI, p. 447, and he has also received a number of patents.*

Before entering into the description of Cowper-Coles' process we wish to call attention to the fact that electrolytic refining of iron has been subject of much research work, partly for practical purposes. In the above-mentioned lecture Cowper-Cowles reviewed some of this work, a more complete synopsis of which will be found in a recently published book by F. Foerster¹ and in a bibliography written by A. Müller.² As yet it seems that electrolytic refining has found practical use only for electroplating purposes, such as steel plating of copper engravings for printing of bank notes at the Imperial mint at St. Petersburg, Russia, where such a method has been used since 1860. It seems that the Langbein-Pfanhauser Werke Aktiengesellschaft, Leipzig-Sellerhausen, Germany, has lately been working along the same lines as Cowper-Coles, on a method suggested by F. Fisher.³

Principles of the Method

The Cowper-Coles method is an *electrolytic refining process*. From pig iron he produces iron sheets or tubes of nearly pure iron. The principles of the method are therefore identical with those of other commercial electrolytic refining processes for the purpose of producing pure copper in ingots, sheets and tubes, as well as silver, gold, zinc, nickel and lead in pure quality. The pig iron is used as anode in a solution of an iron salt. On the application of direct current pure iron is deposited on the cathode, while at the same time a corresponding amount of iron is dissolved from the pig iron anode so that the composition of the solution remains unchanged.

The difficulties to be overcome in the development of such a method are mainly following:

(1) A salt solution must be found suitable as electrolyte, with a sufficiently high conductivity and one which is not fouled too rapidly.

(2) Beside the nature of the electrolyte, the voltage and current density must be correctly chosen in order to get a pure dense deposit in form of an even sheet or ingot without spongy formations and without the actual energy consumption exceeding the theoretical requirement to any appreciable extent.

(3) Electrolytic refining is easier with comparatively pure metals such as copper, silver or gold, than with more impure metals such as iron and zinc, since in the latter case hydrogen will be evolved at the cathode as easily as metal deposited.

The electrolyte used by Cowper-Coles is a concentrated solution of ferrous chloride, containing also iron salts of certain organic acids and an amount of iron oxide is added to give a consistency of a gruel. Additional organic compounds are also used to produce more satisfactory results. The organic acid,

*English patents, No. 20990/06, 21081/06, 21082/06, 28897/06, 28897a/06, 29300/06, 10367/07, 12747/07, 22311/07, 22312/07, 3160/08, 3626/09, 10655/09.

¹"Beiträge zur Kenntniss des electrochemischen Verhaltens des Eisens," pp. 59-81, published by W. Knapp, Halle, Germany, 1909.

²See the journal, "Metallurgie" (Germany) Vol. 6, p. 159, 1909.

³See, for instance, Svensk Kemisk Tidskrift, 1911, p. 135 (Sweden).

of which the iron salt is contained in the solution, is according to a statement in the inventor's former lecture, $\text{CH}_3(\text{OH})\text{C}_6\text{H}_4\text{SO}_3\text{OH}$, consisting of several isomeric cresol-sulphonic acids. The additional iron oxide is intermingled for the purpose of reducing the acidity and polishing the iron sheet which is deposited on a rapidly rotating cathode. The anode used by Cowper-Coles consists of pig iron.

It might be expected that nearly pure iron would be deposited—that is soft sheet iron without carbon or impurities. There is, however, in fact some carbon in the iron obtained by the

of the latter, the thinnest sheets, can most easily be produced.

As to the application of the process for the far more important problem of producing iron of highest purity and quality from ore, it seems that this has been only slightly studied by the inventor, and we shall briefly return to this subject later.

Construction of the Electrolytic Cell

A sketch of the electrolytic cell used at the experimental plant at 20 Danvers street, Chelsea, London, is shown in Fig. 1. The illustration is made partly from sketches and dimensions taken by the authors and partly from a drawing furnished by the inventor. It is drawn at a scale of 1:8, the dimensions being in millimeters.

B is a tank of riveted 5-mm sheet iron containing the electrolyte. Its height is 310 mm and it consists of two concentric cylinders with a bottom. In the horizontal cross section the outer and inner circles show the walls of the tank. On the inside of the tank walls is applied a heavy coating of insulating paint which is of great importance as they otherwise would serve as an anode and rapidly be corroded.

At the inside of the outer wall there are three iron anodes, *A*. They are each 300 mm long and 12 mm thick and are made of curved pig-iron plates, each covering about 1/12 of the circumference. The anodes are separated from the tank through an insulating material, *I*. As will be noted the anodes cover together only about one-quarter of the surface of the tank.

The cathode proper, *K* consists of a 0.5-mm. copper-plated iron cylinder, which is placed outside another cylinder, *C*, of heavier sheet iron (perhaps 5 mm.). The upper and lower edges of the cathode, *K*, are turned out slightly and the space filled with asphalt, probably for the purpose of preventing iron to deposit there. The height of the cathode, *K*, is about 185 mm and its diameter is about 275 mm. The space between cathode and anode is about 40 mm.

The cylinder, *C*, serves as a conductor to *K* and also to support the cathode in such a way as to allow a rapid rotation.

For the latter purpose, the cylinder, *C*, with its sheet-iron cover, *H*, is connected with a vertical steel axle, *E*, supported by bearings *S* and *D*. Through the belt, *R*, the cylinder, *C*, and the cathode, *K*, are rapidly rotated. The shaft, *E*, has just above the pulley a ring attachment, *U*, which during the rotation of the shaft is partly submerged in a bowl filled with mercury, which bowl is not connected to the shaft. The mercury is connected to the negative busbar and transmits current to the rotating cathode *K*.

Finally there is under the tank a circular gas burner for the heating of the electrolyte.

Experiments Performed at the Authors' Visit

The electrolyte used on this occasion was, according to a statement by the inventor, prepared as follows:

Concentrated hydrochloric acid (sp. gr. 1.17), mixed with the cresol-sulphonic acids mentioned before, is heated with a large amount of scrap iron, which is dissolved, while developing hydrogen, until the solution is saturated with iron. The solution, the specific gravity of which has increased as a result of the solution of the iron, is diluted until the specific gravity is 1.4, after which, even on cooling to ordinary temperature no iron salt (ferrous chloride) crystallizes out. The solution contains then about 40 per cent ferrous chloride (FeCl_2) to which are added before using about 300 grams of ferric oxide (Fe_2O_3) per liter, so that it gets about the consistency of gruel.

The addition of this ferric oxide is primarily made for the purpose of reducing the acidity of the solution as far as possible, with proper consideration given to the hydrolysis of the iron salt solution. Moreover, the ferric oxide affects the condition of the finished product. As mentioned before, since the cathode is rapidly rotated and through friction with the iron oxide the sheet or tube deposited on the cathode receives a high finish. In the Elmore process for the electrolytic production of copper tubes the finish is obtained by means of polishing slate of agate applied under pressure and moved up and down in the longitudinal direction of the tube, while in the Cowper-Coles process the same result is obtained by rapid rotation with accompanying friction with the liquid.

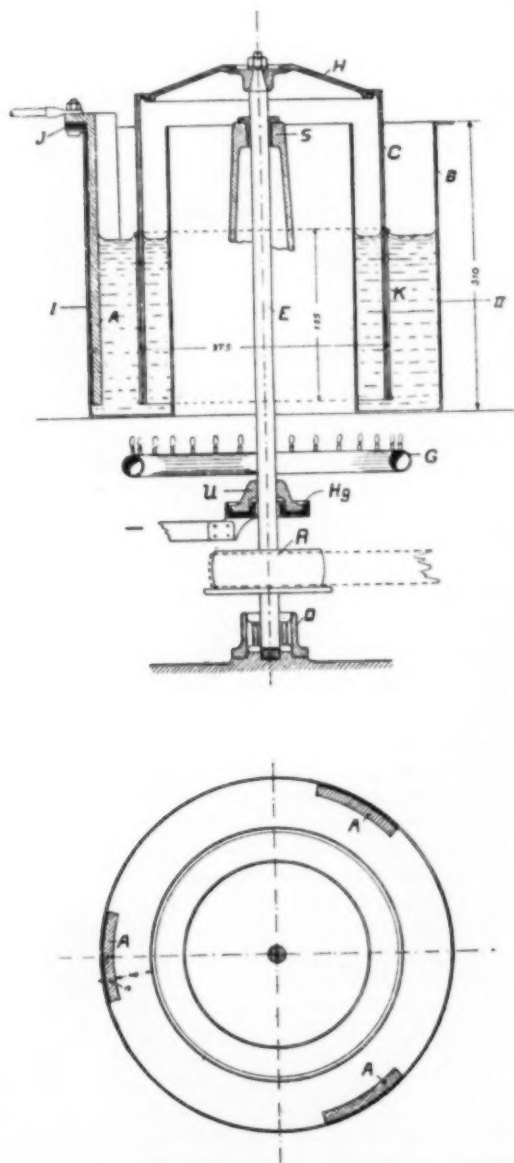


FIG. 1.—CONSTRUCTION OF ELECTROLYTIC CELL

Cowper-Coles process and there are also traces of phosphorus and sulphur.

That carbon can be electrolytically deposited to a small extent, has been stated,* but without endeavoring to control the percentage of carbon, one should be satisfied that the product will always be a soft sheet metal of very high quality. It is also likely that the carbon, which is found by analysis, is due to slime mechanically transported from the anode to the cathode.

According to Foerster (p. 77 of his book) consideration should also be given to the colloidal organic substances which may be deposited on the cathode, as a result of the oxidation of carbon of the pig-iron anode.

On the other hand the electrolytic process has the advantage over the rolling process that the most expensive product

*Coehn, "Zeitschrift f. Electrochemie," 1896, 2, 541.

Experts have testified that an iron sheet produced in a solution containing iron oxide is solid while a sheet produced under other circumstances is perforated by numerous small holes.

The information given by the inventor in regard to the composition of the electrolyte is confirmed by the analysis we have had made at the Material Testing Laboratories of the Stockholm Royal Technical Institute of samples of the solution taken immediately before and after the electrolysis. These tests are referred to as I and II respectively. At first the solution was tested with the ferric oxide contained in it.

	I.	II.
	Before	After
	Electrolysis	
Specific gravity	1.60	1.67
Amount of slime in grams per liter:		
Dried at 105 deg. C.....	245	293
Heated to glowing.....	240	288

The percentage of slime is found to be somewhat different with the amount of stirring at the different times when the samples were taken. Due to the same fact the specific gravity varied somewhat.

Subsequently the filtered solution was analyzed. In both samples it had a light green color, showed a weak acid reaction, and had an evident smell of carboic acid (phenol).

The result of the analysis is as follows:

	I.—BEFORE		II.—AFTER	
	Grams per Cu.Cm.	Per Cent Weight	Grams Per Cu.Cm.	Per Cent Weight
Specific gravity of solution at 150° C.....	1.46	1.46
Bivalent iron.....	0.235	16.1	0.235	16.1
Trivalent iron.....	Trace	Trace
Sulphur as SO ₂	0.0189	1.29	0.0223	1.53
Chlorine.....	0.290	19.9	0.285	19.5
Ferrous chloride, Fe Cl ₂ , calculated from quantity of chlorine.....	0.518	35.5	0.510	35.0
Cresol-sulphonic acid, corresponding to the quantity of SO ₂	0.0445	3.0	0.0524	3.6

The small differences in the analysis should be considered as experimental errors and we conclude that no difference in the composition before and after electrolysis has been established. This was not unexpected, since the time of electrolysis was only two hours.

The amounts of FeCl₂ and cresol-sulphonic acid correspond approximately to the figures given by the inventor.

For the purpose of comparing the energy consumption and voltage of the Cowper-Coles process with that used in other electrolytic refining processes, it was of interest to investigate the conductivity of the electrolyte used. The result is stated below, the conductivity, *C*, being the reciprocal of the resistance, in ohms, of a column of 1 cm length and 1 sq. cm cross section. For comparison we have given some results of measurements of the ordinary electrolyte for copper refining (1 normal CuSO₄ + 1 normal H₂SO₄).

Cowper-Coles Electrolyte		Copper Electrolyte	
Temperature	<i>C</i> .	Temperature	<i>C</i> .
100 deg. C.	0.213	101 deg. C.	0.270
13 deg. C.	0.0506	42 deg. C.	0.231
		13.8 deg. C.	0.179
		0 deg. C.	0.144

According to these and other measurements the conductivity of the Cowper-Coles solution at 100 deg. C. decreases 0.88 per cent per degree.

For the experimental temperature of 92 deg. C. of Cowper-Coles solution *C* = 0.198 and if corrected for the ferric oxide which occupies in volume 58 cu. cm per liter and which is a non-conductor of electricity *C* = 0.187.

In copper refining ordinary or somewhat raised temperatures (45 deg. C.) are used and it seems that at these temperatures the conductivity of the solution is about 0.2 that is nearly the same value as above. The voltage for copper refining is stated

to be 0.25 volt to 0.4 volt, while in the Cowper-Coles process it is about 1.4 volt. This great difference in voltage is not due to the lower conductivity of the Cowper-Coles solution but as calculation shows is due to the greater current density and the greater distance between the electrodes, while the anode surface is smaller than the cathode surface. It may be mentioned that in electrolytic silver refining a still higher voltage is used, namely, 2 volts to 2.5 volts.

The volume of the electrolyte used at our visit may be estimated at about 15 liters.

In calculating the current or current density, the surface of the cathode was taken as 16 sq. dm. as given in the description of the electrolyzer. The voltage varied in electrolysis I between 1.25 and 1.45 volt according to the temperature, the voltage decreasing as the temperature increased. The temperature was read a number of times and varied in electrolysis I between 88 deg. C. and 94 deg. C.

The speed of rotation at the cathode varied in electrolysis I between 108 and 160 revolutions per minute.

In calculating the efficiency it has been assumed that one ampere will deposit 0.0174 gram of iron per minute from the solution of ferrous salts, there being no appreciable amount of ferric salts.

Electrolysis I

In the first electrolysis a cylinder of thin copper-plated sheet iron was used as cathode. To loosen the electrolytically deposited sheet the cathode cylinder had to be heated until glowing.

The electrolysis was carried out for 2 hours, 2.5 min.

Average current 107.9 amp.
Current density at cathode..... 6.74 amp. per sq. dm.
Average voltage 1.34 volt
Average temperature +92 deg. C.

The average speed of the rotating cathode was 139 r.p.m., corresponding to a peripheral speed of 120 meters per minute.

Efficiency.—After heating the cathode in a reducing atmosphere, whereby deposited hydrogen was also removed, the iron sheet could be removed from the cathode and weighed. It was soft and had a thickness of about 0.15 mm. Its weight was 192 grams, not including some pieces which had fallen off nor the iron deposited on the cylinder *C*. This amount (192 grams) corresponds to an efficiency of 83.5 per cent, assuming that 230 grams could be deposited by the number of ampere-hours used.

In order to account for the total ampere-hours,

the iron sheet produced corresponds to..... 83.5 per cent
Hydrogen evolved corresponds to..... 11.8 per cent
Iron lost and deposited on cylinder *C*, escaped
hydrogen, errors in test, corresponds to..... 4.7 per cent

100.0 per cent

Electrolysis II

In the second electrolysis a cylinder of thin zinc-plated sheet iron was used as cathode. From this cathode the deposited iron sheet could be removed without previously being heated. The electrolysis was carried out for 2 hours 30 minutes.

Average current 110.6 amp.
Current density at cathode..... 6.91 amp. per sq. dm.
Average voltage 1.43 volts
Average temperature +94.5 deg. C.

The average speed of the cathode was 145 r.p.m., corresponding to a peripheral speed of 125 meters per minute.

Efficiency.—The sheet, in this case deposited on a zinc-plated cathode, could be removed without heating, but was very brittle and broke into a number of pieces. This was due to the high percentage of hydrogen. The pieces had a total weight of 274 grams, some pieces, however, being missing. In this case it is doubtful whether the sheet was perfectly dry. Furthermore, it was already somewhat oxidized at the time of weighing. After dipping a piece in boiling water it was a little less brittle. After heating in air it was no longer brittle, but rather hard.

The sheet contained before heating a considerable quantity of hydrogen, namely, not less than 0.45 per cent, according to the analysis of the Material Testing Laboratories, of Stockholm, given below. In 274-gram iron sheet this amounts to 1.23 grams hydrogen and its deposition required 11.8 per cent of the coulombs consumed.

This percentage the inventor considers unusually high, but we have decided to figure with the values obtained in Electrolysis II, as well as Electrolysis I, which was performed in the same manner. In heating the sheet the hydrogen (1.23 grams) is removed and the weight of the tempered sheet is thus $274 - 1.2 = 272.8$ grams, corresponding to an efficiency of 94.5 per cent, since the theoretical weight of the deposit is 288.7 grams.

The ampere-hour balance sheet is therefore:

The iron sheet deposited corresponds to..... 94.5 per cent
Deposited hydrogen corresponds to..... 11.8 per cent

106.3 per cent

Oxidation and moisture on sheet and errors.. —6.3 per cent

100.0 per cent

The current density in the Cowper-Coles method is approximately 6.8 amp. per sq. dm. at the cathode (it was in the two electrolyses 6.74 and 6.91 amp per sq. dm. respectively). The highest current density mentioned by Foerster is only 3 to 4 amp per sq. dm. and it seems, therefore, that Cowper-Coles has accomplished a considerable improvement in this respect. A high current density is desirable, even at the expense of a higher voltage, as it has a direct bearing on the time consumed, and thus increases the output of the plant.

Loss in Weight of Anodes.—In test II the total loss in weight at the three pig-iron anodes was ascertained in order to compare how this agreed with the ampere-hour consumption. After the anode slime had been removed the total loss was found to be 291.4 grams, while, under the assumption that the iron is dissolved as well as precipitated as bivalent, it is estimated that the iron taken off the anodes is equal to the amount deposited on the cathode, or 288.7 grams.

However, the anode slime removed weighed 20 grams and the weight of the iron deposited on the cathode was 272.8 grams, and as no increase in weight of the total percentage of iron in the solution had been noticed, it seems reasonable to expect that the difference between loss in weight of the anodes and the amount of iron deposited on the cathode would approximately equal the weight of the anode slime. This difference is $291.4 - 272.8 = 18.6$, which, indeed, nearly corresponds with the actual weight of the anode slime (20 grams).

That the anodes lose more than the weight of the iron deposited on the cathode is customary in electrolytic refining of impure metals and is in the present case due to the following causes:

It is to be expected that the carbon and silicon compounds and possibly sulphur and phosphorus compounds in the pig iron are affected not at all or only slightly by the current and remain as anode slime, which is gradually separated from the anode.

Further, small particles of iron are mechanically separated if the anodes are unevenly consumed.

If the weight of the anode slime is 20 grams, this would be 7 per cent of the iron consumed (219.4 grams). However, it is possible that this percentage of loss is too low.

On one hand it is possible that some slime has been separated from the anode during the operation, in which case the slime removed from the anode and weighed after the test was finished would be too low.

On the other hand, it is also possible, as stated above, that the value of deposited iron (272.8 grams) is somewhat high and consequently the difference between this value and the loss in weight of anodes is also too low.

In copper refining the anode slime is stated to be 4 per cent as a maximum, but the crude copper is considerably purer than the pig iron.

Since the composition of the anode slime, compared with that of the pig iron, is of considerable interest, the slime was analyzed by the Material Testing Laboratories of Stockholm, as well as the pig iron used in these experiments. The result was as follows:

	Pig Iron Used For Anodes	Anode Slime
Iron.....	93.11% (being the difference)	13.5%
Graphite.....	2.51, 2.81% total carbon	23.3, 23.8% total carbon
Combined carbon.....	0.30, 2.81% total carbon	0.5, 23.8% total carbon
Silicon (Si).....	2.30	33.9, corresponding to 15.9% Si
SiO ₂		
Manganese.....	0.43	1.5
Sulphur.....	0.10	1.78
Phosphorus.....	1.23	2.14
Copper.....	0.02	0.16
Chlorine, water, oxygen, etc.....		23.22 (difference)
	100%	100%

From this it is evident that in the anode slime, which still contains some mechanically separated iron, carbon, silicon and sulphur is greatly concentrated, while the phosphorus has been largely oxidized (phosphoric acid) and gone into the solution.

It is interesting to note that the copper is also concentrated in the anode slime and is not dissolved, quite in accord with observations in other electrolytic refining processes. Manganese is also somewhat concentrated, probably due to oxidation to peroxide.

Composition and Properties of the Deposited Sheet Iron

At the Material Testing Laboratories of the Royal Technical Institute, in Stockholm, the iron sheets produced in the tests have been analyzed as to chemical composition and tested as to rusting and mechanical and electrical properties. The analyzed material consisted of:

1. Iron sheet, 0.52 mm. thick, which had been heated and thereby rendered soft. It had been submitted by the inventor and is designated as El. 1 in the following.

2. Sheet produced in electrolysis 2, about 0.15 mm. thick, not previously tempered and consequently hard and very brittle, designated as El. 2 in the following.

A. CHEMICAL COMPOSITION

The results of the analysis are given below, together with the previously given analysis of the pig iron used for the anodes, which shows plainly the result of the refining.

	El. 1. 0.52 Mm. Tempered	El. 2. 0.15 Mm. Not Tempered	Pig Iron
Carbon.....	0.08%	0.06%	2.81%
Silicon.....	0.009	2.30
Manganese.....	0.05	0.43
Sulphur.....	0.02	0.10
Phosphorus.....	0.042	1.23
Chlorine.....	0.18 (=0.32% FeCl ₂)	0.49
Hydrogen.....	0.021	0.45
Hydrogen, after heating to 850° C.....	0.002
Copper.....	0.02

A glance at these analyses shows at once that in regard to carbon, silicon, sulphur and phosphorus very good results are obtained, these elements being present in large quantities in the pig iron, while only small amounts are to be found in the electrolytically deposited sheet. The fact that even if steel anodes are used with a small percentage of carbon, and otherwise as pure as possible, their presence cannot totally be avoided is shown by older observations. The percentages in electrolytic iron are (according to Foerster) as follows:

C.	Si.	S.	P.
0.007-0.089%	0.003-0.01%	0.001%	0.004-0.02%

In regard to manganese it was shown previously by the analysis of the anode slime that it is somewhat concentrated. This indicates, as mentioned, that part of the manganese is oxidized at the anode and does not dissolve. Another part of

the manganese is probably dissolved as manganese chloride. That, nevertheless, such a small quantity of manganese is present in the cathodic iron sheet is probably due to the fact that the manganese being less "noble" than iron is not so readily deposited. It is, therefore, probable that it is concentrated in the solution in the same manner as the iron contained in crude copper is concentrated in copper refining. An older statement* that the manganese is not dissolved with the iron is not confirmed by the Cowper-Coles method.

As already mentioned, the percentage of hydrogen is remarkably high in the non-heated sheet. For information it may be mentioned that a percentage of 0.45 of hydrogen corresponds to not less than 394 units of hydrogen per unit of iron in volume, while the 0.021 and 0.002 per cent of hydrogen in the sheets, which had been heated, correspond to 18.4 and 1.75 units of hydrogen per unit of iron in volume respectively. That electrolytic iron contains a high percentage of hydrogen has long been known, earlier records, however, showing about 0.1 per cent (see Foerster, pp. 64-73). The high percentage in the iron sheet, produced by the Cowper-Coles method, is probably due to the high current density employed in this process.

Since by heating the sheet EL. 1 at 850 deg. C. the percentage of hydrogen decreased from 0.021 to 0.002 per cent, the heating carried out by the inventor must have been done at a lower temperature or for a shorter time. As mentioned several times, the hydrogen makes the iron hard and very brittle and it must therefore be removed by heating.

The high percentage of chlorine is very remarkable and if it cannot be avoided it is very disadvantageous. It may be that the sheet was not well washed before heating or the sheet may have been a little porous and the pores retain some electrolyte, viz., iron chloride.

Therefore the percentage of iron chloride, FeCl_2 , was calculated from the percentage of chlorine in the above table. The fact that the very thin sheet, produced at our visit, which was not washed particularly well, gave a considerably higher percentage of iron chloride (0.88 per cent) than the other sheet (0.32 per cent) indicates that a thorough washing is very desirable. This might also decrease the percentage of sulphur and phosphorus.

As presence of iron chloride seems to promote the rusting of the sheet and also impair the mechanical properties of the iron, it is evidently of great importance to try to prevent the presence of iron chlorides. That there is a possibility of success in this respect has already been pointed out, and it is to be noted that in copper refining chloride has not been found in the cathode copper.

B. RUSTING

Comparative tests were made with sheet EL. 1 and also a soft basic Martin sheet from Avesta (Sweden), 0.55 mm. thick, and with 0.1 per cent C., in the following designated as Av. Two polished pieces of each sheet were suspended in damp air; after three days the electrolytic sheet was considerably more defective than the Avesta sheet.

The explanation might be found in the presence of iron chloride in the electrolytic iron. This salt is deliquescent in damp air and can thus form a solution of the salt on the surface of the sheet, promoting electric local currents, which are believed to cause the rusting.

C. MECHANICAL PROPERTIES

We shall only mention some qualitative tests made with sheet EL. 2 in order to show how the mechanical properties are improved by heating to a higher temperature and consequent removal of hydrogen.

The Testing Laboratory's *bending test*, after heating to various temperatures, was as follows:

A number of pieces of iron sheet were placed in an electric oven which was heated to increasing temperatures. At each 50 deg. rise in temperature one piece was taken out and this was continued until a piece could be bent double and straightened out again without cracking.

After heating to 450 deg. C. the sample broke at first attempt to bend. After heating to 500 deg. C. the sample could be bent to 90 deg., but cracks appeared when bent to 180 deg. After heating to 550 deg. C. the sample bent double without cracking. In straightening out cracks appeared. After heating to 600 deg. C. the sample bent double and straightened out without cracking.

According to these tests the sheet should be heated to 600 deg., at least if time is limited, for improving the sheet.

The effect of heating to higher temperatures (850 deg. C.) is shown in the following quantitative tests:

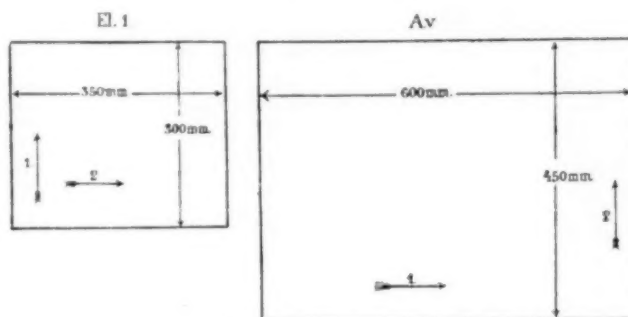


FIG. 2.—TEST PIECES

At these tests was used:

Electrolytic sheet, EL. 1, (Fig. 2) the thickness of which was 0.52 mm and other dimensions 350 x 300 mm.

Martin sheet from Avesta, Av. (Fig. 2), the thickness of which was 0.55 mm and other dimensions 600 x 450.

Strips were taken from these sheets in two directions perpendicular to one another as shown by arrows 1 and 2 in Fig. 2.

Sheet EL. 1 was not made in our presence and we cannot tell with certainty which direction on the deposited sheet cylinder corresponds to 1 and 2. However, it had streaks running parallel with arrow 1.

The samples were tested in their original condition and after heating to 850 deg. C. and cooling they were submitted to stress and bending tests.

The results are given in Table I:

TABLE I.—STRESS TEST

Sample	Direction in Sheet	Physical Condition	DIMENSIONS			Tensile Strength Kg. per Sq. Mm.	ELASTICITY IN PER CENT ON	
			Width, Mm.	Thickness, Mm.	Cross-Section, Sq. Mm.		Unit Length	100 Mm.
EL. 1	1	Not heated	24.8	0.52	12.84	55.9	10.8	7.1
	2	Not heated	24.2	0.52	12.95	35.8	0.5	0.2
	1	Heated	24.2	0.52	12.58	38.2	15.3	11.8
	2	Heated	24.5	0.52	12.74	32.8	2.3	2.0
Av.	1	Not heated	24.7	0.55	13.59	41.0	11.3	8.6
	2	Not heated	24.7	0.55	13.59	33.1	16.0	15.2
	1	Heated	24.6	0.55	13.53	40.3	14.3	12.2
	2	Heated	24.8	0.55	13.64	36.3	13.3	11.3

Bending Test.—In this test samples 20 mm wide were bent over a bar with 2 mm radius, until the break occurred. The results are given in Table II:

TABLE II.—BENDING TESTS

Sample	Direction in Sheet	Physical Condition	Number of Bends to 90° and Reversals (Average of 3 Tests)
EL. 1.....	1	Not heated	3
	2	Not heated	3
	1	Heated	11
	2	Heated	10
v.....	1	Not heated	12
	2	Not heated	28
	1	Heated	18
	2	Heated	20

The results of further tests are given in Table III:

*Burgess and Taylor, "Electrochemical Industry," 1906, 4, 208.

TABLE III

Sample	Direction in Sheet	DIMENSIONS			Elastic Limit Kg. per Sq. Mm.	ELASTICITY PER		Distance of Breaking Point from Nearest End Mark, Mm.
		Width, Mm.	Thick- ness, Mm.	Cross- Section, Sq. Mm.		40 Mm. %	100 Mm. %	
El. 1	Parallel with streaks, . . .	24.7	0.52	12.84	66.8	12.5	10.5	30
	Parallel with streaks, . . .	19.9	0.52	10.35	65.7	6.5	4.1	12
	Right angle to streaks, . . .	24.2	0.52	12.58	65.3	10.0	7.1	52
	Right angle to streaks, . . .	19.8	0.52	10.30	53.7	1.3	1.2	45

D. ELECTRIC PROPERTIES

The electrolytic sheet (El. 1) tempered at 850 deg. C. had at 16 deg. C. a resistance of 0.116 ohm per 1 m. length and 1 sq. mm cross section or 11.6 microhm per cm². With Benedick's formula* the specific resistance of the sheet in non-heated condition (percentage of hydrogen = 0.002 per cent) is calculated to be 11.4 microhms per cm² at "ordinary temperature," which corresponds very nearly with the direct measurements.

In the calculations the chlorine is not considered as it probably exists as iron chloride. It was our intention to investigate also the magnetic properties but shortage of material prevented us from doing so.

It is to be expected that, due to its pure state, the sheet will show very small hysteresis loss. For purposes where this is desirable high electric resistance is usually also wanted, which the electrolytic iron does not possess.

But the electrolytic sheet iron can be made so exceedingly thin (probably about 0.05 mm) that the hysteresis and eddy currents can be brought down to a reasonable amount.

It seems therefore possible that the electrolytic iron can favorably compete with the nickel and aluminium steel which are produced at a thickness of about 0.35 mm by means of rolling for purposes where low hysteresis loss and high electric resistivity is desirable.

Another desirable property of the electrolytic iron is its high permeability. It is stated that an 1/4-hp motor, after having been equipped with poles of electrolytic iron, according to F. Fischer's method, delivered 1 1/4 hp. (See Svensk Kemisk Tidskrift referred to above.)

Some Factors Which Influence the Economical Result of This Process

As stated above the percentage of carbon can be varied, but it can always be assumed that the product obtained by this process contains only a small percentage of carbon, viz., it is a soft sheet or tube. If sheet iron of great length or width is wanted the cylinder must be of very large dimensions. On the other hand tubes, of which we have seen samples, can be made of greatly varying diameter and length and we wish to call attention to the fact that copper tubes have been made successfully by electrolytic processes in a similar manner.

In regard to thickness of material it seems that the inventor has not as yet produced tubes thicker than 2 mm. He did not, however, anticipate any other difficulties in the production of heavier material than possibly that of the weight.

On the other hand in rolling the thin sheet is the final and most expensive product while in the electrolytic process it is the one most quickly and easily produced. The prospects of the process would, therefore, probably lie in the production of thin sheets and tubes of soft steel.

One defect in the process is the method of removing the sheet and heating. In the description of Electrolysis I it was mentioned that the iron was deposited on a copper-plated cylinder, K, which was taken off the cylinder, C, and heated, after which the deposited sheet iron could be taken off and was

soft. This method will undoubtedly be troublesome in making large sheets, as the cylinder, K, would have to be of very large size. In making tubes the method is simpler.

More serious will be the condition of the cylinder, K, when in constant use, which will probably require renewed copper plating. To remove the sheet without previous heating as was done in Electrolysis II is impossible in commercial practice, as the sheet will break into pieces. The inventor made several suggestions for remedying this defect.

Consumption of Pig Iron

As stated above, the consumption of pig iron is larger than the weight of the deposited iron. The weight of the anode slime is according to our observations about 7 per cent of the weight of the consumed pig iron. This estimate, however, may be too low. We shall, therefore, assume a somewhat larger loss—say 1.1 ton of pig iron is required for producing 1 ton sheet iron—without stating, however, that this figure is on the safe side. This 1.1 ton of pig iron should then represent the amount of iron dissolved and deposited plus the anode slime per ton of product.

In this connection it should be pointed out that, disregarding the anode slime, the anodes cannot be completely utilized: when only a portion is left the voltage increases and it must be replaced, while it also happens that an anode is eaten through and a large portion is entirely separated.

The portions remaining can, of course, be collected and recast, which will slightly add to the cost of production. The same applies to the trimmings of the sheet iron. How long the anodes can be used before replacing can only be decided from experience extending over a long time. As a matter of information it may be mentioned that 90 per cent of the anodes used in copper refining can be utilized before recasting.

Energy Consumption

A. FOR ELECTROLYSIS

In the two electrolyses performed in our presence the mean emf. was 1.34 and 1.43 volt respectively or an average of 1.385 volt per cell. This voltage was measured between anode and cathode and to this should be added losses in conductors. This voltage could probably be decreased to some extent by covering the whole inside of tank, B, with anodes. It may, therefore, be safe to figure on a tension of 1.5 volt per electrolyzer.

The ampere-hour efficiency we assumed to be 85 per cent. Assuming on 12 per cent loss in precipitation of hydrogen and chemical losses in removing and trimming the sheets.

For the horsepower year we assume the year to be $350 \times 24 = 8400$ hours as usual in electrochemical work.

Further as 1 amp-hour at 100 per cent efficiency will deposit 1.045 grain iron from a ferrous salt solution and as 1 hp equals 0.736 kw we find that one electric hp-year at the electrolyzer should produce 3.7 tons (metric tons of 2204 lbs. each) of iron sheets or tubes. Or for the production of 1 ton of iron sheet or tube 0.27 electric hp-year is required.

B. FOR MOTOR SERVICE

Electrical energy is also required for the operation of the motors for rotating the cathode, the operation of the pumps, the mechanical treatment of the product and for lighting. Exact information as to the energy required for these purposes cannot be obtained at present, but it can hardly be of great importance compared with the energy requirements for the electrolysis, probably amounting to 1/5 of the latter.

Cost of Labor.—The inventor states, without furnishing further data, that about 50 hours of manual labor is required per ton of products. In this connection it is to be noted that electrolytic processes are practically automatic and require very little attention.

Heating and Maintenance of the Electrolyte.—We did not receive from the inventor any information from which satisfactory estimates can be made as to such cost. The analyses given before show no change in the composition of the solu-

*C. Benedicks, Recherches physique et physico-chimiques sur l'acier au carbon, Upsala 1904, p. 133.

tion due to the electrolysis, with the exception that the amount of slime had increased, which may be due to the difficulty in obtaining uniform samples. From the short investigation we made of the process it is too precarious to draw any conclusions in regard to the life of the electrolyte.

The cost of finishing the author has estimated at about \$1.50 per ton.

Depreciation.—According to figures submitted by the inventor the first cost—exclusive of power station and iron supply—amounts to approximately \$100 per ton of iron sheet produced per year, buildings amounting to about \$1,600. Allowing 5 per cent for depreciation of buildings and 10 per cent for other items the annual amortization amounts to $\$0.80 + \$8.40 = \$9.20$. The inventor figures only with an amortization at \$4.60 which seems to be too low. This is mentioned only to show that the cost of amortization is a rather large factor, principally due to the high cost of the electrolyzer.

Interest on Investment in Stored Pig Iron.—Electrolytic refining processes operate rather slowly. In Electrolysis II a 6-mm sheet was deposited in 1 hour with the current density used by Cowper-Coles and to produce a 10 mm sheet would require about 7 days of 24 hours. In connection with such processes consideration must, therefore, be given to the interest on stored raw material.

In electrolytic copper refining the interest on the amount invested in raw material is of very great importance and this has been the primary reason for endeavoring to increase the speed of operation as much as possible, that is, operate with highest possible current density. In the Cowper-Coles process the current density is higher—since in copper refining it hardly exceeds 4 amp. per sq. diameter—and beside this the raw material is of comparatively small value, and, therefore, the interest on the amount invested in raw material in iron refining is of no consequence. The inventor has estimated that with an annual production of 5000 tons, a permanent supply of 1000 tons would be sufficient, which is probably ample, judging from the experiences of copper refining. If the value of 1000 tons of pig iron is \$16,000—the interest at 6 per cent would be \$960—or about 19 cents per ton of production.

The Application of the Method for Reduction of Iron Ore

In his lecture before the Iron and Steel Institute Mr. Cowper-Coles also mentioned the application of his method for producing iron sheet and tubes of high quality from iron ore. The inventor has not, however, as yet developed his method for this purpose. On account of the great importance of this matter we wish to add a few words on the subject.

The method which would first suggest itself for realizing the purpose would be to dissolve the ore by acids and extract the iron electrolytically.

In regard to the solubility of iron ores we wish to point out that siderite and bog ore dissolve relatively easily in acids while magnetite and hematite require heating with concentrated sulphuric acid or concentrated hydrochloric acid.

For dissolving 1 ton hematite about 2.76 tons of pure H_2SO_4 are required. On account of the hydrolysis of the iron salts more acid is required than the equivalent weight, for which allowance has been made and 5 per cent has been added; 2.76 tons pure sulphuric acid corresponds to 4.4 ton chamber acid with 62.0 per cent H_2SO_4 . The price of chamber acid is about \$5.50 per ton, possibly less. The cost of only the acid will thus be about \$24 per ton. For magnetite the cost is 8/9 of this or about \$21.50.

With hydrochloric acid the cost is considerably higher. It is, therefore, evident that this application of the process will be commercially possible only if the acid can be regenerated, which is a possibility. In this case carbon or some other inert substance should be used as anode.

Furthermore, for the regeneration of the acid a diaphragm would have to be used to separate anode and cathode. Under this circumstance and as long as the cathode consists of a rotating cylinder, the apparatus will be rather complicated.

If hydrochloric acid is used the regeneration could be ac-

complished by passing the chlorine, which is developed at the anode from the iron chloride solution, together with steam over incandescing coke.

In using hydrochloric acid there would be added, however, a considerable expense for evaporating the water absorbed in the circulating process. This might also occur in connection with sulphuric acid.

On account of the necessity of using insoluble anodes instead of soluble pig-iron anodes the voltage would be higher. From observations in other electrochemical processes it may be estimated that not less than 5 volts would be required. Assuming an efficiency of 85 per cent as before hematite ore would yield per hp-year 0.73 ton or 1 ton iron would be deposited by 1.37 hp-year. Magnetite would yield 0.82 ton per hp-year and 1 ton of iron would be deposited at the expense of 1.22 hp-year.

In this case, soft steel, practically free from carbon, would be produced, unless some new startling innovations are made, and the amortization of the expensive plant would be quite heavy.

At the present state of developments we shall not venture to express any opinion as to the future possibilities of the process with respect to its application to the direct reduction from iron ores.

Stockholm, Sweden.

The Outcome of the Carborundum Patent Suit

The patent law suit of many years between the Electric Smelting & Aluminum Company and the Carborundum Company has now reached what would appear to be its final stage, by a decision rendered on March 15 in the United States Circuit Court of Appeals for the Third Circuit.

The case was tried before Circuit Judge Gray and District Judges Bradford and Young. The decision is written by Judge Bradford. A dissenting opinion was handed down by Judge Young.

From the decision written by Judge Bradford we quote the following:

"These appeals have been taken from a final decree of the Circuit Court of the United States for the Western District of Pennsylvania made Feb. 21, 1911, confirming the report of a master, ¹ an accounting of profits for patent infringement ordered by that court, in this case Dec. 7, 1900. The original suit for infringement was brought by the Electric Smelting & Aluminum Company, hereinafter called the complainant, against the Carborundum Company, hereinafter called the defendant, for infringement of three United States patents, No. 319,795 for 'Processes for Smelting Ores by the Electric Current,' No. 319,945 for 'Electric Smelting Furnaces' and No. 335,058 for 'Electric Furnaces and Method of Operating the Same.'

"The two first-named patents were issued to Eugene H. Cowles and Alfred H. Cowles June 9, 1885, and the last-named patent to Alfred H. Cowles Jan. 26, 1886, and all of these patents were assigned to the complainant. At the final hearing in the Circuit Court the charge of infringement of Patent No. 335,058 was abandoned, and that court held that neither of the other two patents had been infringed.

"On appeal to this court the decree of the Circuit Court was in May, 1900, affirmed as to Patent No. 319,945, but reversed as to Patent No. 319,795; it being held by this court that claims 1, 2 and 4 of the last-named patent had been infringed by the defendant, and the case was referred by the court below pursuant to the decree of this court to a master for an accounting of profits derived by the defendant from the infringement.

"The defendant thereafter and while the case was in the hands of the master adopted, Jan. 1, 1901, what it claimed to be a different and non-infringing process, and in the language of its counsel, 'did it almost on the instant,' and claimed that it was not bound to account for any profits derived from its manufacture and sale of carborundum after Dec. 31, 1900, as at that time it discontinued using the process held to have been infringed. This claim the complainant denies, contending that

the alleged new process was a clear infringement of Patent No. 319,795.

"The master decided in December, 1906, that the alleged new process was not an infringement of that patent, limited the accounting to Dec. 31, 1900, and made an award of \$22,411.62 in favor of the complainant, all of which was confirmed by the court below.

"Before taking up the question of the sufficiency of the sum awarded to the complainant as profits during the accounting period up to and including Dec. 31, 1900, we shall briefly consider whether the defendant was not thereafter guilty of infringement of the complainant's process for which it should have been compelled to account.

"We think the court below was in error in holding that the process employed by the defendant from and including Jan. 1, 1901, to June 9, 1902, the date of the expiration of Patent No. 319,795, was not an infringement of that patent, and that the complainant was not entitled to profits gained by the defendant from the use of that process during that period. This court in and by its decree in 1900 reversed the decree of the court below and decided that claims 1, 2 and 4 of that patent were valid and had been infringed by the defendant. These claims are as follows:

"1. The method of generating heat for metallurgical operations herein described, which consists in passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore to be treated by the process being brought into contact with the broken or pulverized resistance material, whereby the heat is generated by the resistance of the broken or pulverized body throughout its mass, and the operation can be performed solely by means of electrical energy.

"2. The method of smelting or reducing ores or metalliferous compounds herein described, which consists in subjecting the ore in the presence of carbon to the action of heat generated by passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore being in contact with the broken or pulverized resistance material, whereby the ore is reduced by the combined action of the carbon and of the heat generated solely by the resistance of the broken or pulverized body throughout its mass.

"3. The method of smelting or reducing ores or metalliferous compounds herein described, which consists in subjecting the ore in the presence of a reducing agent to the action of heat generated by passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore being in contact with the broken or pulverized resistance material, whereby the ore is reduced by the combined action of the reducing agent and of the heat generated solely by the resistance of the broken or pulverized body throughout its mass.

"The claims of the patent should be liberally construed for reasons heretofore given by this court in this suit, as follows: 'On careful examination we have failed to find in any patent, publication or other matter alleged as an anticipation or as showing the prior art, a practical process for metallurgical or analogous operations involving the use of a discrete body of conductive but resistant material rendered incandescent by the passage of an electric current and mixed or otherwise in contact with the material to be treated. This is the broad underlying idea of the process patent in suit, and is well covered by its claims. The Messrs. Cowles were the first to invent and use this process and the patent must be sustained. It is a meritorious one and its claims are entitled to considerable liberality of construction.'

"This court has declared in this case in relation to the core used by the defendant: 'The use of the central core of resistance material is within the process of the appellant under claims 1, 2 and 4. These claims require merely contact in contra-distinction to a mixture between the resistance material and the material to be treated, and it is immaterial whether, under those claims the body of resistance material assumes the

form of a central core or any other shape, so long as it is in contact with the material to be treated, is discrete, granular or pulverized in its composition, is rendered incandescent through the passage of electric current through its mass or over its area, and thereby affords the heat to effect the desired end.'

"And further: 'If claims 1, 2 and 4 are broad enough to include actual contact of any kind between the two materials, the resistance material forming part of the electric circuit may be in contact with the material to be treated, whether in the form of a core or several cores, or of strata, or in any other form, to secure the most effective operation of the process under varying conditions involving the nature of the material to be treated, amount of the charge, etc.'

"And in the description of the patent it is stated: 'The scope of this invention is not limited by the degree of granulation, as that may vary with the conditions of the case, and with a large furnace and a powerful current the size of the carbon particles may pass beyond what is ordinarily understood by the term granular and be in fact pieces of carbon of considerable size.'

"The defendant in its alleged new process employed a block core consisting of a number of blocks of electric light carbon, each 32 in. long and 4 by 4 in. in thickness. They are arranged in 'zig-zag' fashion and the ends of each two blocks forming a V rest upon a carbon block from 12 to 14 in. long and 4 by 4 in. in thickness, called a connector block. A large quantity of fine granular or pulverized carbon, or graphite, is disposed between the carbon blocks and around their ends on the connector blocks, and also a large quantity of granular or crushed coke is disposed at each end of the furnace to convey the electric current to and from the electrodes to the block core, that core being in electrical connection with the electrodes at each end of the furnace.

"Notwithstanding the furnace in the arrangement of the resistance material we are satisfied on the evidence that the alleged new process practised by the defendant since Dec. 31, 1900, is the process, the use of which by the defendant this court held to infringe claims 1, 2 and 4 of Patent No. 319,795; the mere mechanical difference in the form and arrangement of the core not serving to affect or change the nature of the process or to shield the defendant from responsibility. The defendant must, therefore, be required to account to the complainant for all profits gained by the former from its manufacture of carborundum under the alleged new process, and its sale, from and including Jan. 1, 1901, to June 9, 1902."

The opinion then takes up in detail the manner in which the sum awarded to the complainant had been arrived at by the master. The total net profits during the accounting period had been \$104,525.74.

"The master deducted from the \$104,525.74, representing total profits, 49.55 per cent thereof, or \$50,792.51, for 'profit properly belonging to the invention of the defendant of the product, carborundum itself,' and 'the profit belonging to the invention of defendants of the bonding material, that under the evidence made the great bulk of the sales of carborundum possible by the defendant company.' After deducting from the total profits the above sum of \$50,792.51, the master apportioned to the complainant, but subject to further deduction, the remaining 50.45 per cent, amounting to \$52,733.23.

"This apportionment was arrived at by ascertaining that the cost of the entire output of the defendant's business during the period of infringement was \$469,665, and that the cost of making carborundum, including its cleaning, crushing, grading, and preparation for sale, was \$236,944.53, or 50.45 per cent of the total cost of the defendant's output, and by assuming that 'each dollar of expenditure by the defendant earned an equal amount of profit.'

"He then proceeded to deduct from the above sum of \$52,733.23 15 per cent thereof as representing the difference between the 'whole cost of carborundum production' and the 'furnace cost,' being \$7,909.99, leaving a balance, subject to further reduction, of \$44,823.24 as 'the part of the profits to which the use of complainant's process in the production of

carborundum contributed, after consideration of the foregoing elements as entering into that profit."

"The master next proceeded to deduct one-half of the above sum of \$44,823.24 as the amount of the supposed contribution to the defendant's profits through its ownership of an Acheson patent and its re-issue for carborundum granted subsequent to the issuance of Patent No. 319,795 in suit, and awarded to the complainant the remaining half, namely, \$22,411.62 as the total amount it was entitled to receive under the accounting ordered by the court below pursuant to the decree of this court, together with the costs of accounting. . . .

"The complainant contends that in addition to such sum as may be decreed to it for profits made by the defendant from its infringement of Patent No. 319,795 during the period from Dec. 31, 1900, to June 9, 1902, the complainant is entitled to recover from the defendant the above sum of \$104,525.74 without deduction. This contention is based upon two grounds: first, that whatever profit there is or has been in the defendant's business has been derived solely through its infringement of the complainant's process, and, secondly, that, if it be assumed that the first ground is untenable, there has been such negligence on the part of the defendant in failing to keep accounts showing the amount of profit derived by it from the infringement that the complainant is entitled to recover the total profits of the defendant's business.

"The first of these grounds we think is well taken. It clearly appears from the evidence that the business of the defendant is based upon and practically confined to the manufacture and sale of carborundum in one form or another. As stated by the court below in its opinion (189 Fed. 710) as to a proper form of decree to be made under the decision of this court on final hearing (102 Fed. 618): 'The respondent has a large plant built and adapted for the manufacture of carborundum. The machinery and apparatus were likewise built for that special work. It is respondent's sole business, and in that article and its application they have built up a large trade and extensive use.'

"The making of wheels, cloth, paper, etc., covered in whole or in part with carborundum is wholly subsidiary or incidental to its advantageous sale. Admittedly it would be impossible for the defendant without its wrongful use of complainant's process to carry on its carborundum operations. Not only could it earn no profits, but its business would cease to exist; for no other process for the manufacture of carborundum than that of the complainant has been disclosed.

"Much has been said on the subject of the Acheson process and product patents No. 492,767, of Feb. 28, 1893, and re-issue No. 11,473, of Feb. 26, 1895, for 'Improvements in the Production of Artificial Crystalline Carbonaceous Materials.' In the description in both of these patents Acheson states: 'My invention consists in the various processes and products of said processes substantially as hereinafter more particularly set forth. . . . I do not limit myself to any of the particulars set forth, as I intend to cover and claim broadly the new crystalline carbonaceous materials whether produced by the particular processes or methods set forth, or by any substantially equivalent methods, and for all purposes to which it may be applied.'

"And his claims were as broad as his statement of invention. It is contended in substance by the defendant that the patent in suit, while covering the process of manufacturing carborundum, does not cover the article itself; that Acheson was the original inventor and discoverer of the article; that the defendant, as his assignee, has a monopoly in that article however or wherever or by whomsoever produced in this country; and, therefore, that the carborundum sold by the defendant in the conduct of its business, although manufactured in defiance of the patent in suit, must be treated as a contribution by the defendant towards profits realized and be considered on the accounting in question.

"A valid process patent, though for a process only, confers on that patentee a right to exclude all others from using such process for the attainment of its object. Others may secure

the desired result, if they can, by the employment of a different process, if open to them, but they cannot without leave or license of the patent owner use the patented process with impunity. And on an accounting for profits, and not for damages, in a case of infringement, where profits to the infringer are impossible save through his infringement, he must be treated as a trustee *ex maleficio* and can withhold none of his gains from the patentee. The fact that in such a case the patent is not for the product, but only for the process, is wholly immaterial; for the infringer will not be permitted by a court of equity to take advantage of his own wrong, but will be held accountable as a trustee of the profits.

"It is well settled that, in the case of mechanical inventions, where the infringed patent covers a mere improvement upon mechanism before known and open to the defendant to use, the complainant can recover only the excess of such profits as have been realized through the use of the improvement over what the defendant might have made by the use of such mechanism without such improvement. But it is equally well settled that where the entire commercial value of the mechanism arises from the patented improvement the owner of the patent will be entitled to recover from the infringer the total profits derived from the manufacture, use or sale of such mechanism. . . .

"Nor is the application of the rule affected in a case where profits, and not damages, are to be accounted for by the fact that the owner of the patent has not used the mechanism containing such patented improvement. *Crosby Valve Company vs. Safety Valve Company*, *supra*. The principle of the foregoing cases, involving the consideration of mechanical inventions, is not without much weight in the determination of the question of profits in the case before us. We think it applies *a fortiori* here, as it was only through the infringement of the complainant's process that profits were made, or indeed, were possible. For, as above stated, no other process than that of the complainant is disclosed under which carborundum can be manufactured.

"Further, the holder of a valid process patent has a right not only to manufacture under the patented process, but to use and sell the product, unless that product be the subject of a valid patent monopoly in favor of another person. The defendant contends that under and by virtue of the Acheson patent it has a monopoly in the product carborundum. If it had such a monopoly it would seem to follow that, save in so far as equitable considerations might produce a different result, it would have the exclusive right to use and sell carborundum, whether wrongfully manufactured by itself or rightfully manufactured by the complainant; and in the latter case under the process patent upheld by this court as valid and meritorious, while the complainant would have the exclusive right to manufacture carborundum under its process, it would be powerless to use or sell a pound of such manufactured product. It is not perceptible that under such circumstances the process patent would have any value.

"A clear case should be made to justify the court in treating the defendant's product patent as operative or enforceable against the complainant and thus practically nullify the process patent so far as it covers the manufacture of carborundum. Acheson did not discover, invent or have any idea of carborundum until long after the patent in suit was granted.

"On final hearing not only his patents, but much evidence touching his connection with the product, were before this court when it was said: 'We have no doubt on the evidence that carborundum can be and has been produced under the process covered by any of the claims of the process patent in suit, although not so efficiently or profitable under the method of claim 3. So long ago as 1884 the Messrs. Cowles produced it under their process with and without the use of a prepared or pre-formed central core of resistance material. It is true they did not manufacture it for commercial purposes and, failing to appreciate its importance, though aware of its abrasive qualities, applied their process to other branches of metallurgy.'

"And, further: 'The Messrs. Cowles were the first to invent and use this process and the patent must be sustained.'

"It appears from the evidence that, although carborundum was not manufactured on a commercial scale before the Acheson product patent was obtained, specimens of carborundum made by the Messrs. Cowles and the Cowles Electric Smelting & Aluminum Company, the assignor of the complainant, were placed on exhibition in different parts of the country and much notoriety was given through lectures and printed publications to the incandescent feature of the patent in suit. The Messrs. Cowles, it is true, failed to appreciate the importance of the product, as above stated, and did not know it by the name of carborundum—for it had yet not been so christened—and were not familiar with its composition, but they were aware of its highly abrasive qualities.

"There is no evidence that the Messrs. Cowles or their successors in the title to the process patent in suit ever intended to abandon that process so far as the manufacture of carborundum is concerned. The fact that they exhibited from time to time and in different places specimens of what is now known as carborundum strongly indicates an intention on their part not to exclude that product from the operation of the process patent. This court has decided that the process patent in suit is valid, meritorious, and entitled to considerable liberality of construction.

"This case presents some unusual features. Under the process Patent No. 319,795 the Messrs. Cowles and the Cowles Electric Smelting & Aluminum Company had a right, fully sustained by this court to manufacture carborundum, and they also had a right—certainly until the Acheson patent was obtained—though not necessarily an exclusive right to use and sell any carborundum made by them. In fact, they did make carborundum under the patented process, though not on a commercial scale. As before stated, there is no evidence that they or the complainant ever abandoned the right to apply the patented process to the manufacture of that article. The Acheson patent and its re-issue, which were obtained a number of years after the patent in suit, claimed not only carborundum but also a process for the production of that article. This court has decided that the defendant in using the process claimed by Acheson used the process of the patent in suit and was guilty of infringement. It does not appear that there was any other process under which carborundum could be produced.

"If the Acheson patent so far as it claims the product in contradistinction to the process should be held to confer upon the defendant the exclusive right to use and sell carborundum, then, as before indicated, the complainant's process patent was held absolutely at the mercy of the defendant; for the bare right to manufacture without a right to use or sell what is manufactured is valueless.

"While it has repeatedly been stated that a patent may be obtained for a process in contradistinction to its product, or for a product in contradistinction to the process under which it is created, we can recall no case in which it has been held that after a valid process patent has been granted and applied to the production of an article, though not on a commercial scale, and there has been no abandonment by the patent owner of the right to apply the process to the production of the article on such scale, and such article can be produced only under that process, another person can obtain a patent conferring upon him an exclusive right, as against the holder of the process patent, to the use and sale of the article so produced and thereby wholly destroy the value to him of the process patent so far as it may be applied to the production of such article.

"If such a product patent can validly be obtained it would be necessary in order to avoid gross injustice to the owner of the process patent that all profits rendered possible only through the infringement of the process patent should be awarded to the latter, and to that end the infringer should be treated as a trustee *ex maleficio* of the profits.

Patents are not granted to serve as instrumentalities for the perpetration of fraud or the doing of injustice. They are

property, but, like other property, must be used legitimately. We perceive no reason why the aphorism *sic utere tuo ut alienum non laedas* is not applicable to the use of a patent as well as of other property. To permit one who holds a subsequent product patent to infringe a prior process patent, without which process the product could not be created, and then shield himself in whole or in part from liability for his wrongdoing by setting up his product patent, would not accord with the principles usually obtaining in courts of equity. Under such circumstances the least that equity demands is that the wrongdoer be compelled to disgorge his ill-gotten gains.

"But it is not necessary that we should in this case express an opinion upon the validity or invalidity of the Acheson product patent or its reissue. For whether the defendant did or did not hold a valid patent for carborundum as a product is, as has appeared, immaterial to the decision of the question of profits; for profits having been rendered possible only through the infringement of Patent No. 319,795 the whole amount under the doctrines of the authorities above cited must be awarded to the complainant. . . .

"The decree of the court below must be reversed, with costs to the Electric Smelting & Aluminum Company in this court and the court below, and this case be remanded to the court below with directions that the Electric Smelting & Aluminum Company recover from the Carborundum Company upon a further accounting in the court below the total profits realized by the latter from its manufacture of carborundum and its sale from and including Jan. 1, 1901, to June 9, 1902, together with legal interest thereon from the proper date; and further, that the Electric Smelting & Aluminum Company recover from the Carborundum Company the above-mentioned sum of \$88,743.30, together with legal interest thereon from Feb. 21, 1911; and that such further proceedings be had in the court below as shall dispose of this case in accordance with the views herein expressed. A decree may be prepared and submitted."

Judge Young, in his dissenting opinion "concurs in the foregoing opinion so far as it finds the Carborundum Company guilty of infringement of the process patent in suit during the period from and including Jan. 1, 1901, to June 2, 1902; but dissents from such opinion so far as it holds that the profits realized by the Carborundum Company from its infringement of said patent, together with interest and costs, should not be apportioned between the parties in accordance with the decree of the court below, but awards the total profits, interest thereon, and costs to the Electric Smelting & Aluminum Company."

Barium sulphate precipitated in the determination of barium is likely to be contaminated with occluded alkali salts. Owing to the fine state of subdivision of the precipitate it is difficult to wash these salts into the filtrate. Gooch and Hill, (*Amer. Journ. Sci.*, March, 1913), have found that the precipitate may be made coarsely crystalline and easily washed by dissolving the sulphate in sulphuric acid and then evaporating to dryness by directing the flame of a blast lamp directly onto the solution. In order to avoid loss by spattering, a platinum gauze cone is inverted in the mouth of the crucible. The operation is more rapid than evaporation by a Hempel burner, the time required being about 15 minutes, as compared with from one-half hour to several hours by other methods.

The second annual report of the Director of the U. S. Bureau of Mines, for the year ended June 30, 1912, has just been issued. Among the tangible benefits resulting from the Bureau's work are, improvements in mine safety and in mining conditions, laws and regulations for mine safety, and fuel investigations. The general adoption of permissible explosives, having a short and relatively cool flame, has resulted in reducing accidents. A great deal of money has been saved to the government by specifying grades and quality in fuel purchases. Among the needs of the Bureau are suitable buildings, grounds and equipment for carrying on experimental work at Pittsburgh, extension of mine-rescue and first-aid work, extension of investigations on mine accidents, extension of investigations on mineral waste and metal mining industries.

"Exothermic Steel."

Discussion Before the Pittsburgh Section of the American Electrochemical Society

The third meeting of the Pittsburgh section of the American Electrochemical Society was held at the German Club on the evening of February 19, at the conclusion of an informal dinner. The chairman of the section, Mr. C. G. Schleudersberg, presided.

In the preceding meeting Mr. **Walter O. Amsler** had spoken on the "exothermic steel" process (this journal, vol. X, p. 559, September, 1912) and the present meeting was devoted to two reports on experiments made with this process by Mr. **W. W. Clark**, chief chemist of the American Vanadium Company, and Mr. **Philo Kemery**, metallurgical engineer of the Crescent Works of the Crucible Steel Company of America.

Mr. Clark first presented his report as follows:

Report by Mr. W. W. Clark

At our last meeting Mr. Amsler gave us a little talk on his so-called new process of making steel and its product "exothermic steel." The paper had previously been published in *METALLURGICAL AND CHEMICAL ENGINEERING* (vol. X, p. 559), where it received considerable criticism from the different iron and steel metallurgists throughout the country (vol. X, pp. 712, 713, 774, 775, 776).

To give a short synopsis of Mr. Amsler's talk, he takes a mixture of iron ore, feldspar, lime and bauxite, which he heats to 1800 deg. Fahr., or about 1000 deg. C. in a graphite crucible. At this temperature a violent reaction is supposed to take place, with sufficient evolution of heat to raise the temperature of the entire mass above the melting point of the steel, while all the iron oxide is reduced to the metal.

He first claimed that the feldspar and bauxite did the trick, and in his article had a composition formula for feldspar, which, if it were possible, would reduce some iron. He also had the reaction which took place between the iron oxide and the feldspar, showing how the iron was reduced.

He admitted that he did not know anything about this formula, as he was not a scientific man at all, but had just assumed the formula to suit the case, but said that if it were that way it would reduce some iron, but we know from analysis and from all the known laws of chemistry and metallurgy, that it is impossible for feldspar to take up oxygen. He later admitted that maybe feldspar did not do the work, and that carbon entered into the reaction, but still insisted that he was making steel that way, and would be glad to show one or all of us, how it was done.

He also stated that the steel had been in use for some time at the Bureau of Mines in the shape of cannon for the testing of the explosive power of the different powders. Upon investigation of this steel, the Bureau of Mines did not know anything about exothermic steel, but said that they had bought steel from Mr. Fogler, who had a shop in the west end of Pittsburgh, and that it was very good steel, but they were unable to say how it was made. The cannon came to them ready to use.

Taking Mr. Amsler at his word, I offered him the use of our laboratories to test out his process in the presence of two or three of the eminent steel metallurgists of the Pittsburgh district, and although since that time, I have telephoned and written Mr. Amsler, I have been unable to get him to come.

A number of experiments were made, following Mr. Amsler's directions given at our last meeting with negative results, and also some following new directions sent by Mr. Amsler in a letter to me which reads as follows:

"I am at the present time very busy with contract work which must be gotten out at a certain date, but if I can possibly arrange it I will come out to your laboratory some time next week. In making your experiments in a small crucible, try using 6 or 8 oz. of ore, $\frac{1}{2}$ oz. of bauxite, 1 oz. of feldspar and $\frac{1}{2}$ oz. of lime. In heating this charge be sure to use a reducing flame, or cover in such a manner that the oxygen

from your fire cannot enter the crucible. This is essential as otherwise the oxygen absorbing properties of the flux will be satisfied with the free oxygen, rather than that which is in combination with the iron in the ore. I believe under these conditions you will be able to show practically all of the iron in the ore in a metallic state. (Signed) Walter O. Amsler."

A No. 4 Dixon graphite crucible was used in each case where carbon was present, while an alundum crucible was used for the experiments in the absence of carbon. A platinum-platinum rhodium couple and a Siemens-Halske galvanometer were used to record the temperatures. When carbon was present carbon fire ends outside quartz were used, while in the experiments without carbon, the couple was protected by magnesite. The crucibles were placed inside a larger crucible, the space between packed with pulverized coke, and tight covers placed on both. A small hole was bored through both of the covers for the insertion of the thermo-couple.

Experiment 1.—Mix: 6 oz. pure iron oxide, $\frac{1}{2}$ oz. bauxite, 1 oz. feldspar and $\frac{1}{2}$ oz. lime.

The material was carefully weighed and mixed, and then ground to 80 mesh. The crucible was placed in a larger crucible, packed with coke, charged with the above mix, the covers adjusted, and the whole transferred to the furnace which had previously been heated to 1000 deg. C. The mix was held at this temperature for 3 hours. The melt was pasty and after cooling showed small globules of iron scattered throughout the mass with the majority around the edge, where it had come in contact with the carbon of the crucible. The whole melt was weighed and showed by analysis that about 7.5 per cent of the iron had been reduced. Temperature variation 106 deg. C.

Experiment 2.—Mix: Same as in experiment 1.

Brought to 1000 deg. for 1 hour and then raised to 1250 deg. for $2\frac{1}{2}$ hours. The melt was fluid and a small amount of iron had been reduced. This by analysis showed 13 per cent of the iron reduced. Temperature variation at 1000 deg.—64 deg. C. and at 1250 deg.—94 deg. C.

Experiment 3.—Mix: Same as in experiment 1. Brought to 1450 deg. for 3 hours. Melt was very active and poured clean. The iron had been melted and settled to the bottom. This by analysis showed 28 per cent of the iron reduced. Temperature variation 103 deg. C.

Experiment 4.—Mix: Same as in experiment 1. Melted in an alundum crucible. Held at 1250 deg. for 3 hours. The melt was fluid and after cooling showed fine homogeneous structure. By analysis this showed 0.29 per cent of the total iron reduced. Temperature variation 47 deg. C.

Experiment 5. Mix: Same as in experiment 1. Melted in an alundum crucible. Brought to 1450 deg. C. for 3 hours. The melt was very fluid but quiet. This gave 0.38 per cent of the iron reduced. Temperature variation 41 deg. C.

Experiment 6.—Mix: 6 oz. iron oxide, $\frac{1}{2}$ oz. fluorspar, 1 oz. cryolite and 1 oz. soda ash. Melted in a graphite crucible. Held at 1000 deg. for 1 hour and then brought to 1250 deg., where it was held for 2 hours. Showed 11.4 per cent of the iron reduced. Temperature variation at 1000 deg.—82 deg. C., at 1250 deg.—61 deg. C.

Experiment 7.—Mix: Same as in experiment 6. Brought to 1450 deg., and held for 3 hours. This showed 31.4 per cent of the iron reduced. Temperature variation 45 deg. C.

Experiment 8.—Mix: Same as in experiment 6. Melted in an alundum crucible. Brought to 1250 deg. C. for 3 hours. Showed 0.63 per cent of the iron reduced. Temperature variation 23 deg. C.

Experiment 9.—Mix: Same as in experiment 6 with the addition of 1 oz. 80-mesh charcoal in an alundum crucible. Held at 1250 deg. for three hours. Showed 86.3 per cent of the iron reduced, but was not melted. Temperature variations 94 deg. C.

Experiment 10.—Mix: Same as in experiment 1 with 1 oz. charcoal added. Held at 1250 deg. for three hours. The melt was more fluid than in experiment 9 and showed 88.8 per cent of the iron reduced, but the iron was not melted. Temperature variation 85 deg. C.

Experiment 11.—A quantity of the mix of experiment 1 without the iron oxide was melted in a platinum crucible in an atmosphere of hydrogen, cooled and weighed. The weight of the material was 21.6076 grams. It was again melted and held for three hours in an atmosphere of oxygen, cooled and weighed. The weight was 21.5937 grams, showing a loss, instead of an increase in weight, as might have been expected from Mr. Amsler's talk and his letter.

From these experiments the following conclusions are drawn: That the mixture of feldspar, bauxite, and lime, as recommended by Mr. Amsler, does not facilitate the reduction of iron from its oxide; that it does not absorb oxygen in any way, nor does it help carbon or other reducing agents to absorb oxygen in any way, except by means of its fluxing action. That any other easily melting flux has the same effect. That any reduction which takes place is entirely due to the action of carbon, and not due to any properties of the mix as reported. That the reaction is not exothermic as reported; in fact it does not show any appreciable change of temperature, during the melt, and that without the carbon no iron is reduced, while if carbon does the reducing, the reaction is endothermic. The total effect of the mix is only one of a flux, and not one which is easily handled. It seems to me that Mr. Amsler has taken this process on faith and not on actual practice as he claims.

Mr. Amsler's process is not a process at all, and I for one am very sorry that the patent laws of our country are so loose, that they will permit the patenting of such a thing, that on the face of it is impossible, and which on trial proves to be a pipe dream, and the exploiting of such things for the innocent investors, who think that if a thing is patented, it is all right.

In the discussion of Mr. Clark's paper Mr. Amsler objected to the phrase that "the exothermic steel process was his process." Mr. Amsler said that Mr. Fogler and not he was responsible for the process. He could not explain why the reactions did not take place in Mr. Clark's experiments. During the discussion Mr. Amsler also quoted Dr. Munroe, of George Washington University, as vouching for the process.

Mr. Philo Kemery, metallurgical engineer of the Crucible Steel Company of America, then presented his report on his own experiments, as follows:

Report by Mr. Philo Kemery

I have made ten experiments with the "exothermic steel" process, following Mr. Amsler's instructions as given in his talk at our former meeting. My experiments with results briefly stated, follow:

Experiment 1.—36 lb. hammer scale, 1 lb. feldspar, $\frac{1}{2}$ lb. bauxite, 1 lb. lime, were all thoroughly mixed and placed in a regular steel melting crucible. The temperature was brought up to a good yellow heat in a small heating furnace. Results: no reaction.

Experiment 2.—The crucible from Experiment 1 with its contents was then placed in a crucible furnace along with a heat. At the end of one hour it was noticed that the crucible was cutting badly, due of course, to the scale and the fluxes. At the end of two hours the crucible was withdrawn, the bottom portion still held together while the top crumbled in the process of drawing. In this experiment 11 lb. of iron was reduced which was to be expected since years ago hammer scale and iron ore were reduced in the same way without the use of the above fluxes.

Experiment 3.—In this experiment, iron ore was substituted for hammer scale as it is much more readily reduced.

80 oz. iron ore, 4 oz. feldspar, 2 oz. bauxite, 4 oz. lime were well ground, thoroughly mixed and placed in a small plumbago crucible. At the suggestion of Mr. Amsler it was heated in a round cast-iron stove using pulverized coke as fuel. He directed that the coke be piled around and on the top of the crucible so as to insure a reducing atmosphere which was followed out.

Some two or three tests were tried, the temperature being brought up to a good yellow heat for from two to three hours.

In all there was found some reduced iron, but in no case was there any evidence of an exothermic reaction. The volume of the contents of the crucible was reduced to about one-half the original in nearly all cases.

Experiment 4.—Later, it was further suggested by Mr. Amsler that a little carbon, say, about 10 per cent, facilitated the reaction, so a new mixture was made up as follows:

No. 1.—40 oz. iron ore, 20z. feldspar, 1 oz. bauxite, 2 oz. lime, 4 oz. charcoal.

No. 2.—40 oz. iron ore, 2 oz. lime, 4 oz. charcoal.

All were ground and thoroughly mixed. Several parallel runs were made by heating in the stove. Crucible No. 2 showed about the same as No. 1. Both contained reduced iron in the form of a spongy mass, but no evidence of any exothermic reaction.

Experiment 5.—Parallel runs with the above mixtures were then made in a heating furnace in which the temperature was raised almost to a white heat with negative results. Beginning with experiment 4 parallel crucibles were run in which No. 1 contained the active fluxes, feldspar and bauxite, while in No. 2 they were left out for comparison of results.

Experiment 6.—Since the heating furnace did not produce as good a reducing flame as is thought necessary, several parallel experiments were tried in a blacksmith's forge using the same formulas as in experiment 4.

Here again the coke was filed all around and over the top of the pot and in 40 minutes the temperature brought up to the melting point of steel.

Crucible No. 1 showed a spongy mass of more or less reduced iron. A glassy slag was formed. The contents were not sufficiently melted to teem. In crucible No. 2 there was less slag, the other conditions being about the same as in No. 1.

The volume was much reduced, the residue or reduced iron from No. 1 could not be differentiated from that of No. 2. No exothermic reaction.

A crucible of steel scrap was then placed in the forge and heated for 40 minutes as near as possible under the same conditions. When removed the pieces of scrap were all fused together. A $\frac{1}{4}$ -in. iron rod was melted at this temperature in about 1 minute, which shows that the temperature was much above that where the exothermic reaction is reported to begin, namely, 1000 deg. C.

The "matte" from experiment 6 was heated in an open-hearth furnace, the crucible being placed on the bank just inside the door of the furnace and examined by tilting the crucible every five minutes. At the end of 35 minutes the contents was boiling. When teemed on the floor 11 grams of metallic iron was separated from the slag. A crucible of steel scrap placed alongside the above melted in 25 minutes.

Experiment 9.—Another crucible of "matte" was heated in the open-hearth furnace at a slightly lower temperature for 35 minutes. A good slag was formed, but when the crucible was teemed a semi-reduced mass of iron was left in the bottom.

The crucible of steel scrap was partially melted.

Experiment 10.—In this experiment three crucibles were used. No. 1.—40 oz. iron ore, 2 oz. feldspar, 1 oz. bauxite, 2 oz. lime, 4 oz. charcoal.

No. 2.—40 oz. iron ore, 2 oz. lime, 4 oz. charcoal.

No. 3.—All steel scrap.

All three were placed in an open-hearth furnace at same time. No. 1 in 55 minutes showed a fine glassy slag, but when teemed there was left the same spongy mass of semi-reduced iron.

No. 2 in 35 minutes developed a little slag with a semi-reduced residue in the bottom that could not be distinguished from that of No. 1. No. 3 was melted in 25 minutes.

These crucibles were tilted over every five minutes for the purpose of examination.

In all of these experiments the crucibles were closely watched, for it is claimed that when the content of the crucible is raised to about 1000 deg. C. the chemical reactions within develop sufficient heat to raise the temperature to that of melted iron. It

is further claimed that when the exothermic reaction begins the content of the crucible is likely to boil over. In no case did I see anything that resembled or approximated these claims.

In a majority of these experiments the contents of the crucible was much reduced in volume. A crust of semi-reduced iron was formed on the top and all around the side the



ONE OF THE PRODUCTS OF EXPERIMENT 10 OF MR. KEMERY

thickness of which depended upon the time and temperature. The results further show that the oxides of iron were reduced about as well without the use of feldspar and bauxite as with. The products of experiment 10 were exhibited to the society in which No. 1 could not be distinguished from No. 2.

General Discussion

Following Mr. Kemery's report Mr. Amsler stated that he himself did not believe at first Mr. Fogler's contentions that he could reduce iron ore to steel by means of his flux. Mr. Amsler re-quoted from his first paper: "This steel was first brought to the writer's attention about a year ago by Mr. George Fogler, the inventor. At that time the claims and descriptions of the process appeared rather absurd and the statements of results were hard to believe. Close investigation of this steel already in use and of the reactions and the theoretical side of the operation led to the conviction that the process has considerable merit." Mr. Amsler again quoted Dr. Munroe, of George Washington University, as endorsing the process.

Mr. Kemery then asked Mr. Amsler whether in the melting this "matte" in an open-hearth steel furnace any difficulty was experienced in the handling of the slag.

Dr. Schluederburg, chairman of the Pittsburgh Section, asked Mr. Amsler if Mr. Fogler would not come down to Mr. Clark's or Mr. Kemery's laboratory and be present while a laboratory experiment was being made.

Mr. Amsler replied that Mr. Fogler was adverse to laboratory experiments and placed no reliance on them. Mr. Amsler said that the material was produced commercially and quoted a case where the Braburn Steel Company had made some Otis elevator shafts in one case where eleven pots of about 100 lb. had been required for one cast.

Mr. Amsler was asked if this flux had not been used some ten years ago in the reduction of copper from its ore.

Mr. Amsler replied that this flux would also apply to copper.

Mr. Kemery reported that in the case where anything like

steel was produced in his experiments, the steel obtained ran 0.134 in sulphur which alone would condemn the steel produced. This sulphur undoubtedly was obtained from the feldspar.

Mr. Amsler questioned as to why the Braburn Steel Company would not furnish a sample of steel for test purposes replied that the Braburn Steel Company did not have a license to make this material and hence would not make this steel or sell this steel to anyone outside of Mr. Fogler.

We are indebted to Mr. Sydney Cornell, secretary of the Pittsburgh section, for the above report of the discussion.

The Willard Gibbs Medal for Dr. Baekeland

The Chicago Section of the American Chemical Society has elected Dr. Leo H. Baekeland, of Yonkers, N. Y., to be the recipient of the Willard Gibbs Medal founded by William A. Converse. The first award was made in 1911 to Prof. Svante Arrhenius, director of the Nobel Institute, at Stockholm, Sweden. The second medalist was Prof. Theodore W. Richards, of Harvard University.

The formal presentation of the Willard Gibbs Medal will be made to Dr. Baekeland at the May meeting of the Chicago Section of the American Chemical Society.

Dr. Baekeland is at present in Europe, but is expected to return early in April.

The jury of award which selected Dr. Baekeland comprised Prof. Alexander Smith, Dr. W. R. Whitney, Dr. E. C. Franklin, Prof. W. A. Noyes, Dr. J. D. Pennock, Prof. G. B. Frankforter, Prof. John H. Long, Prof. Julius Stieglitz, Mr. William Brady, Mr. E. B. Bragg, Mr. S. T. Mather and Dr. G. Thurnauer.

Iron and Steel Market

March developed no material change in the iron and steel markets. The situation has indeed been rather colorless since the first of the year, a definite pace being established and showing no material variation. In brief the history of the finished steel trade has been as follows: Production has been maintained at capacity, but operations were particularly good in March, on account of weather conditions, making March the record tonnage month in the history of the industry. New buying, represented by the closing of contracts, has been at from 60 to 90 per cent of capacity, perhaps averaging 75 per cent. Specifications received against existing contracts have uniformly exceeded shipments, though by only a negligible margin. Realized prices on shipments have slowly increased, through the closing out of old contracts, and in March were only about \$2 a ton below the openly quoted prices on new business.

The course in the pig-iron market has been altogether different from that in the steel market. While the advance in steel prices practically ceased early in October, leaving the market stationary thereafter, pig-iron prices continued to advance until about the middle of December, and only a fortnight or so later evidences of weakness began to appear, these developing into a definite decline which continued through the first three months of this year, making the total decline from the high point in December to date between 75 cents and \$1 a ton, though by no means equally distributed. In foundry iron the declines have averaged fully \$1 a ton, while Bessemer iron has not declined at all, and basic iron has declined only 50 cents.

Thus there has been a divergence between steel prices and pig-iron prices, and again a divergence in the movement of different grades of pig iron, but the latter divergence fits with the former; in other words the strength of the steel market has prevented steel making iron from declining much in sympathy with foundry iron.

The physical situation explains the market movement. Through the building of many furnaces by steel works in recent years the steel works have become more self-supporting, as to pig iron, than ever before. The merchant furnace capacity nevertheless has been increased, chiefly by the building of

furnaces by Lake Superior ore interests, seeking a more certain outlet for their ore than sales in the season to season market. The demand for steel, moreover, has for many years increased more rapidly than the demand for iron castings. The gradually increasing demand last year for all classes of material brought the steel industry to a point in October at which it was operating practically full, but there remained many idle furnaces. Some of these furnaces have been blown in, so that from October to March there was practically no increase in steel works production of pig iron, but an increase of 15 to 20 per cent in merchant furnace production. It is probable that there is now surplus production of merchant pig iron, but this is not certain. The decline in the market could be explained, in part at least, by the conduct of buyers, who are in the habit of buying farther and farther ahead if the market is advancing, but who the moment any weakness develops determine to buy no more iron until their contracts and stocks are entirely exhausted. Thus buyers have been playing a waiting game, sufficient to cause a decline in so open a market as pig iron even were the statistical position wholly in favor of producers. Similar movements do not occur in steel partly because pig-iron markets are local, whereas nearly all steel products are sold on a Pittsburgh basis and partly because the steel interests normally work more in harmony with each other than do producers of pig iron. Pig iron contracts are usually enforced, while steel contracts usually are not enforced if the market goes against the buyer. Hence the steel maker is very anxious to see quoted prices maintained, while the furnaces are relatively indifferent.

Pig Iron

Demand for pig iron throughout the country was very light in March, the chief activity being in low-grade iron for pipe foundries in the East and in basic iron in the Central West. Prices declined in almost all markets, Eastern foundry iron losing 50 cents, and valley foundry iron as much or more. The Southern market dropped 50 cents, to \$13, Birmingham, and there are reports that even that figure has been shaded. Lake Erie furnaces west of Buffalo showed the greatest weakness, quoting prices to distant points, not normally in their territory, such as to undermine prices made by furnaces in such territory. Thus Cleveland furnaces offered iron in Pittsburgh, although at a 50-cent freight disadvantage compared with the valleys, having \$1.40 freight to Pittsburgh against 90 cents for the valleys. Consumers of foundry iron, instead of being tempted by reduced prices, are more firm in their attitude of holding off. In several districts increased inquiry is reported, but such inquiry is obviously for the purpose merely of securing information as to the extent of the drop, consumers being quoted lower prices and yet not buying. In the Pittsburgh-valley district sales of about 30,000 tons of basic iron were effected in March, prices declining slightly from sale to sale, until \$16, valley, was reached for third quarter delivery, against a price of \$16.50 maintained from the beginning of December to the middle of February. The market is now quotable as follows: Bessemer, \$17.25; basic, \$16; No. 2 foundry and malleable, \$16.25; forge, \$15.75, f.o.b., valley furnaces, 90 cents higher delivered Pittsburgh; No. 2 X foundry, delivered Philadelphia, \$17.75; basic, delivered eastern Pennsylvania, \$17.50; No. 2 foundry, f.o.b. Buffalo furnaces, \$16.50; at Cleveland furnaces, \$16; at Chicago furnaces, \$17.25; at Birmingham, \$13. Freights from Birmingham are \$3.25 to Cincinnati, \$4.35 to Chicago, and \$4.90 to Pittsburgh.

Steel

No improvement has occurred in shipments of billets and sheet bars on contracts. Mills having adequate contracts are fairly well supplied, but mills indifferently covered by contract are unable to secure extra tonnage in the market. The United States Steel Corporation has continued to be short of steel, and during March purchased 6000 tons from Butler for consumption at Ellwood City, also taking 15,000 tons in the East and South. The market for such odd lots as are occasionally sold continues to be quotable at \$29 for billets and \$30 for sheet

bars, f.o.b. maker's mill, Pittsburgh or Youngstown. Prices charged on long-term requirement contracts are considerably less, but are advancing. The Carnegie Steel Company on March 20 named prices for second quarter on quarterly adjustment contracts \$1 a ton higher than were named for first quarter.

Finished Steel

There have been further decreases in premiums on finished steel for early delivery, and there is now relatively little of such business except in bars, for which product premiums have rather been tending to increase. For early delivery there are now rarely premiums on sheets, and premiums on plates in reasonable tonnages do not run above \$1 to \$3 a ton at the outside. The East mills are obtaining large premiums on steel bars, sometimes \$4 a ton or more, but in the Central West there are hardly any mill sellers for prompt shipment, and jobbers are obtaining an unusually large amount of business.

In steel manufactures more irregularities have appeared. Rivets have been shaded so much that the quotable market is down \$2 a ton and the same can be said of bolts and nuts. There are slight irregularities in merchant pipe, and prices for fabricated steel on small jobs are considerably lower than in January.

Regular prices at Pittsburgh, unless otherwise stated, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Steel hoops, 1.60 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.60 to 1.65 cents, Philadelphia; 1.57½ to 1.60 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.35 cents; galvanized, 28 gage, 3.50 cents; painted corrugated, 28 gage, 2.55 cents; galvanized corrugated, 28 gage, 3.55 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., 80 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 70 per cent off list.

Standard railroad spikes, 1.85 cents.

Button head structural rivets, 2.10 cents; cone-head boiler rivets, 2.20 cents.

The reduction of stannic oxide to tin can be accomplished by molten metallic zinc, according to a patented process of Zdenko Metzl, of Rouen, France. When metallic zinc is melted in a reducing atmosphere and treated with stannic oxide, tin is reduced rapidly and quantitatively, and zinc oxide is volatilized.

The Dorr Cyanide Machinery Co., Denver, Col., has just issued a new catalog descriptive of the three Dorr machines, classifier, continuous thickener and agitator. The booklet contains 36 pages, and is illustrated with installations of the different machines in various parts of the world. Practical data also are given regarding the operation of the machines, and the results obtained by them. The classifier and continuous thickener are better known than the agitator, the last being a new development.

The launder type of Richards pulsator classifier is the subject of a new Bulletin, No. 1050, just issued by the Denver Engineering Works Company, Denver, Colo. This is a comparatively new development in classifiers, and consists of a settling pocket which is bolted to the launder, a sorting column under the pocket, a hutch below the sorting column and the Richards pulsator valve. It is a single compartment classifier, and one is used for each table in the mill.

The behavior of nitroglycerine when heated has been investigated by Snelling and Storm, (Technical Paper No. 12, U. S. Bureau of Mines), who find that the substance begins to decompose at 50 to 60 deg. C. At 70 deg. C. the commercial product gives off nitrous acid after 15 to 20 minutes. Boiling occurs at 145 deg. C., due partly to decomposition and partly to volatilization. Explosion takes place at about 218 deg. C.

Synthetic Ammonia

An account of the Report of Professor Haber which led the Badische Company to take up the Process on a Large Scale

The report on "the technical production of ammonia, from its elements," which was submitted by the inventors, Prof. Fritz Haber and Dr. R. Le Rossignol, to the Badische company three and a half years ago, has now been published in the *Zeitschrift für Elektrochemie* of Jan. 15, 1913 (Vol. 19, p. 53).

Since the report was submitted the Badische company has developed this process on a large scale and in his paper before the International Congress of Applied Chemistry last September (this journal, September 12, Vol. , p. 637, 1912) Dr. H. A. Bernthsen stated that the "problem has now been solved fully on a manufacturing scale and that the walls of the first factory for synthetic ammonia are already rising above the ground at Oppau, near Ludwigshafen-on-Rhine."

Since the report of Haber and Le Rossignol contains a great many technical details which were not given in the paper of Bernthsen, we give here as full an abstract as possible.

Professor Haber and Dr. Le Rossignol first point out that from a commercial viewpoint the production of ammonia from its elements is an exceedingly attractive proposition. The world's demand of ammonia is several hundred thousand tons a year and these are supplied almost completely from nitrogen in coal. In form of 25 per cent ammonium sulphate the ammonia is worth 22.25 cents per kilogram, while the cost of 14/17 kilogram of nitrogen is 1.125 cents and the cost of 3/17 kilogram of hydrogen is 4.375 cents (1 kilogram of ammonia containing 14/17 kilograms of nitrogen and 3/17 kilograms of hydrogen).

In order to determine the best conditions of the synthesis, that is, especially the most favorable temperature and pressure, two points are of chief importance: first, the thermodynamic equilibrium, and, second, the reaction velocity. The thermodynamic equilibrium determines how far at the given temperature and pressure the reaction between the nitrogen and hydrogen gases, forming ammonia, will proceed and where it will stop, and it determines how great the percentage of ammonia is in the final gas mixture of nitrogen, hydrogen and am-

monia, in which the constituents are in equilibrium with each other. The reaction velocity determines how much time will be required between the hydrogen and nitrogen in order to reach the final condition of equilibrium. It is important to note that while the thermodynamic equilibrium depends only on temperature and pressure, the reaction velocity which also depends on these two factors may further be influenced to a very large extent by the use of catalytic agents.

To develop a practical process the conditions of temperature and pressure should, of course, be so chosen as to

yield an equilibrium in which the percentage of ammonia in the final gas mixture is as high as possible. On the other hand, they should be so chosen that the reaction velocity between nitrogen and hydrogen should be as high as possible. But these two conditions are contradictory to some extent. This will be obvious if an attempt is made to use ordinary atmospheric pressure and to select the most favorable temperature at which the reaction should be carried out. If a low temperature is chosen the thermodynamic equilibrium is favorable, since a relatively large amount of ammonia can exist in contact and in equilibrium with hydro-

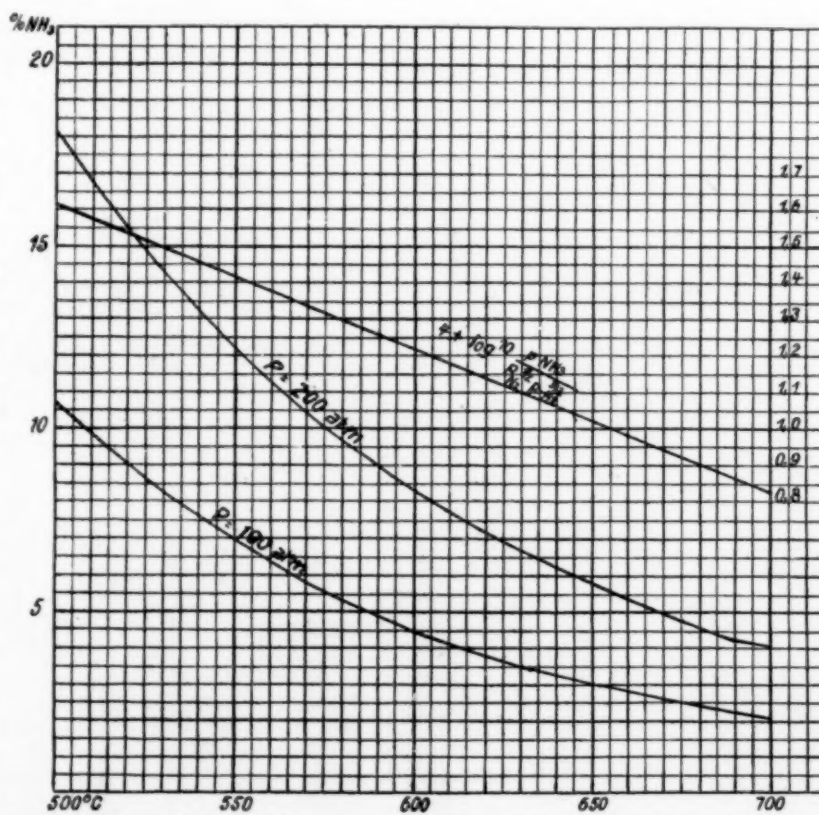


FIG. 1.—THERMODYNAMIC EQUILIBRIUM CONDITIONS OF SYNTHESIS OF AMMONIA

gen and nitrogen gases, but in this case the reaction velocity is exceedingly low since practically the two gases do not react with each other at all and no catalytic agent has so far been found which would be efficient at temperatures of 300 deg. C. or less. On the other hand, if a high temperature is used the reaction takes place quickly but stops very soon since the equilibrium is reached when only a very small amount of ammonia is formed.

It is evident that to solve the problem ordinary atmospheric pressure cannot be used, but the best conditions of both pressure and temperature have to be selected. Practical considerations have led Haber and Le Rossignol to the conclusion that the temperature should be chosen between 500 and 700 deg. C. If a higher temperature than 700 deg. C. would be chosen the conditions of the thermodynamic equilibrium become unfavor-

able. If temperatures below 500 deg. C. would be chosen the reaction to be complete would require too much time and too large an amount of catalytic agent. Within the limits of 500 and 700 deg. C. the conditions of the thermodynamic equilibrium are given in Fig. 1.

This diagram gives in the straight line the equilibrium constant and in the two curves marked $P=200$ atmospheres and $P=100$ atmospheres, the percentages of ammonia in the final gas mixture which represents thermodynamic equilibrium. For instance, with a pressure of 200 atmospheres the final gas mixture in which its constituents are in equilibrium contains 18 per cent ammonia at 500 deg. C., 8 per cent ammonia at 600 deg. C. and 4 per cent ammonia at 700 deg. C.

While these percentages are satisfactory, they are not high enough to conduct the process in such a simple way that the gas would be compressed, passed over the catalizer, whereupon the ammonia formed would be absorbed, and the balance of the gas would be wasted. The waste would be too great. The process must, therefore, be conducted in such a way that after the ammonia is recovered from the final gas mixture, the remaining gases, together with a fresh supply of H and N gases, are subjected again to the action of the catalytic agent. This problem has been solved by Haber and Le Rossignol by devising a partial cyclic process for the movement of the gas, the ammonia being formed at a high temperature and the ammonia being removed from the gas mixture in form of liquid ammonia at a low temperature, the remaining gas mixed with fresh gas being then subjected again to the action of the catalytic agent, and so on.

Care must be taken not to use too low a temperature for the removal of the ammonia from the gas mixture, since there

Practical experience with liquid air indicates that "regeneration of heat and cold" becomes particularly efficient and easy when using high-pressure gases. The same has been found to be true by Haber and Le Rossignol in their experiments.

FURNACE CONSTRUCTION

Both quartz and metallic containers were used in their experiments. Above 700 deg. C. and with pressures up to 50 atmospheres Haber and Le Rossignol used quartz tubes, but did not dare to use them at 100 and 200 atmospheres. Later on when they found that it would be advantageous to use temperatures below 700 deg. C., Haber and Le Rossignol developed various metallic furnaces, several types of which were heated from the inside by electric resistance wire. Their construction is described in detail and illustrated in the report.

Later on it was found possible to use so low a temperature that it was sufficient to use iron containers heated from the outside. The continuous production of ammonia became possible by means of a simple sort of test tube of steel, shown in Fig. 2. A steel cylinder with 6 mm. inside diameter and a 4 to 5 cm. length contains the catalytic agent. The inside space slants outside at the top, forming a cone, so that a steel stopper can be pressed into it, also of conical form. The angle of the cone of the stopper (16 deg.) is less than the angle of the cone in the cylinder head (20 deg.). By turning the screw as shown in the illustration the apparatus can be closed absolutely air tight.

The nitrogen and hydrogen gases are introduced through a steel capillary tube which reaches down to nearly the bottom of the reaction chamber. The gaseous products of the reaction leave the chamber through another capillary tube at the top.

About 2 or 3 grams of osmium are used as catalytic agent in this apparatus. To heat it, the whole apparatus is placed in a fused saltpeter bath and the nitrogen-hydrogen gas mixture is passed under high pressure slowly through the apparatus. With a bath temperature of 555 deg. C. and a pressure of 83 atmospheres the percentage of ammonia in the gases passing out of the apparatus was 5 per cent, which is very nearly equal

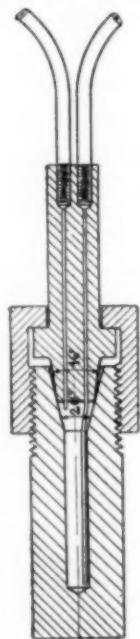


FIG. 2.—SIMPLE FURNACE FOR SYNTHESIS OF AMMONIA

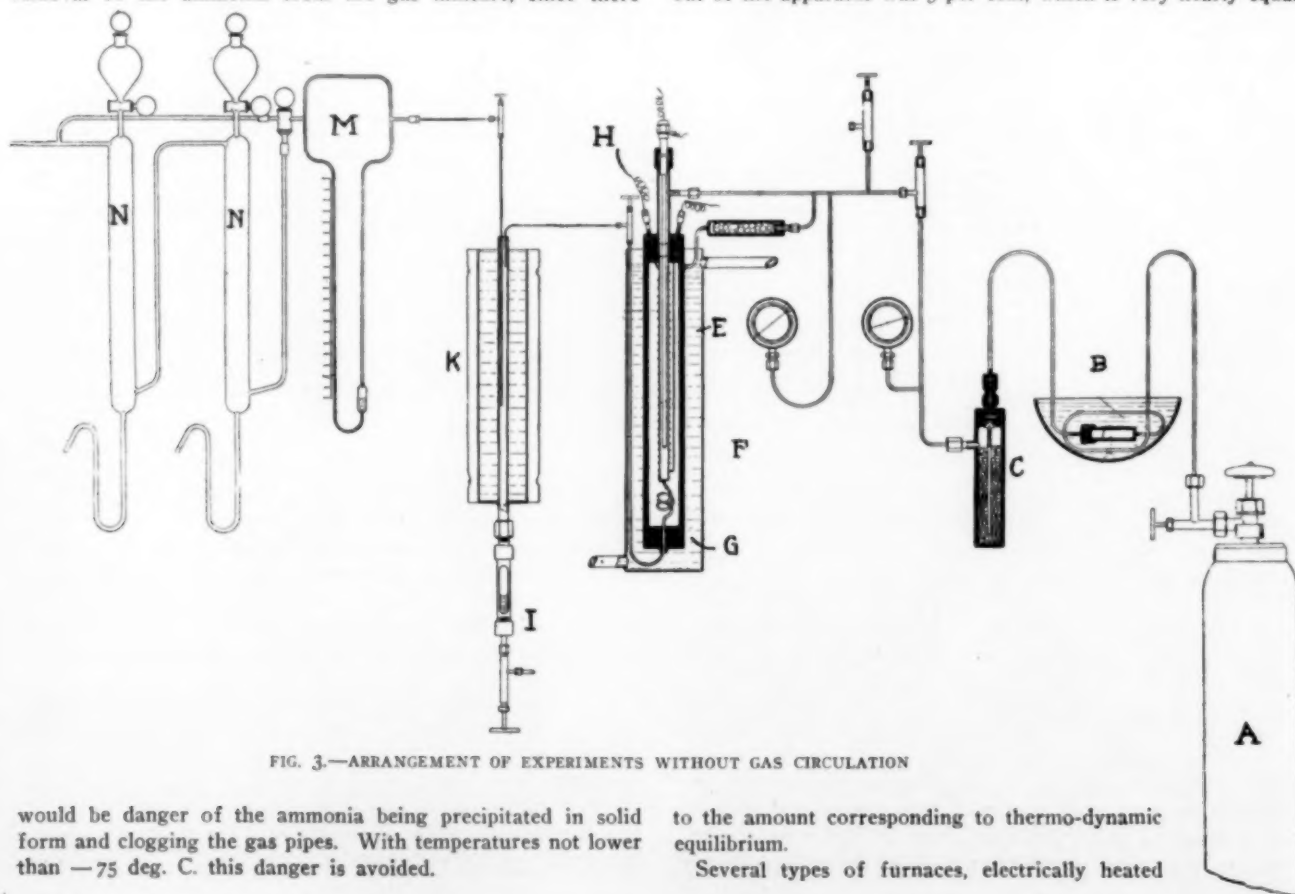


FIG. 3.—ARRANGEMENT OF EXPERIMENTS WITHOUT GAS CIRCULATION

would be danger of the ammonia being precipitated in solid form and clogging the gas pipes. With temperatures not lower than -75 deg. C. this danger is avoided.

to the amount corresponding to thermo-dynamic equilibrium.

Several types of furnaces, electrically heated

from the inside, are described, the construction being given in considerable detail and in various diagrams in the original report. Especially if electric heating is employed it is important not to lose the heat, but to regenerate it. The use of heat regeneration has also the advantage that the gases which leave the catalyzer at a high temperature have a low temperature when they reach the apparatus exit.

The method of heat regeneration employed by Haber and Le Rossignol is very simple, the gas mixture being forced to pass through prescribed passages, the heat regenerator con-

In the second series of experiments the technical practicability of the whole process as a continuous process had to be tested in that the gas mixture coming from the furnace had to be cooled, the ammonia removed, and the remaining gases returned to the furnace, etc., in continuous operation.

The arrangement of the first series of experiments (without gas circulation) is shown in Fig. 3. The mixture of hydrogen and nitrogen gases is passed from the steel container *A* through a copper capillary tube into an iron cylinder *B* filled with palladium asbestos and heated in a saltpeter bath. On account of the combination of the last traces of oxygen in the gas with hydrogen some water is formed which is removed by passing the gas through calcium chloride *C*. The gases then pass to the furnace *E* in which *F* is the thermo-element, *H* the heating wire, and *G* the cooling water.

The mixture of hydrogen and nitrogen gases and of the ammonia formed in the furnace *E* passes out through a thin steel capillary tube into the iron tube *K* which is subjected to cold so as to liquefy the ammonia. To the iron tube *K* is connected at its bottom a glass tube *I* of thick walls in which the liquefied ammonia accumulates and can be measured and drawn off at will.

This refrigerating apparatus *K* for the liquefying of the ammonia was not used in all experiments. In the majority of the experiments the liquefier *K* was disconnected and the gas mixture passed from the furnace *E* directly into the meter *M* which measured the velocity of the gas. The determination of the ammonia in the gas mixture could be carried out in a great many cases by an optical method (apparatus *N N*) by using the Rayleigh gas interferometer of Carl Zeiss, according to the method proposed by Haber for the analysis of furnace and mine gases.

The arrangement of the second series of experiments (with gas circulation) is shown in Fig. 4. The hydrogen and nitrogen gas enters at *A* into the furnace *F* in which the ammonia is formed, and the ammonia, nitrogen and hydrogen gas mixture leaves the furnace through the capillary coil *B*, which is embedded in

cooling water. The gases pass two valves, pass through the dryer *C* (where any traces of water are removed) and then enter the "cold regenerator" *D*. This consists of a system of three copper capillary tubes wound as shown in the detail sketch *X*. The gases having been strongly cooled in the regenerator, enter the liquefier *D* in which the ammonia is liquefied. The level of the liquid ammonia can be seen at *E*.

The liquid ammonia may be drawn off through *F*. The hydrogen and nitrogen gas mixture from which most of the ammonia has been removed by liquefaction, is then returned to the cold regenerator *D* and is then forced through the pump *P* back into the furnace *F*, while a valve permits to introduce new hydrogen and nitrogen gases simultaneously into the furnace.

All the constructional details of the pump and the other high-pressure apparatus were worked out by Professor Haber and Dr. Le Rossignol themselves and the apparatus were built by them with the aid of the machinist of their laboratory. Since doubts had been expressed concerning the practicability of working

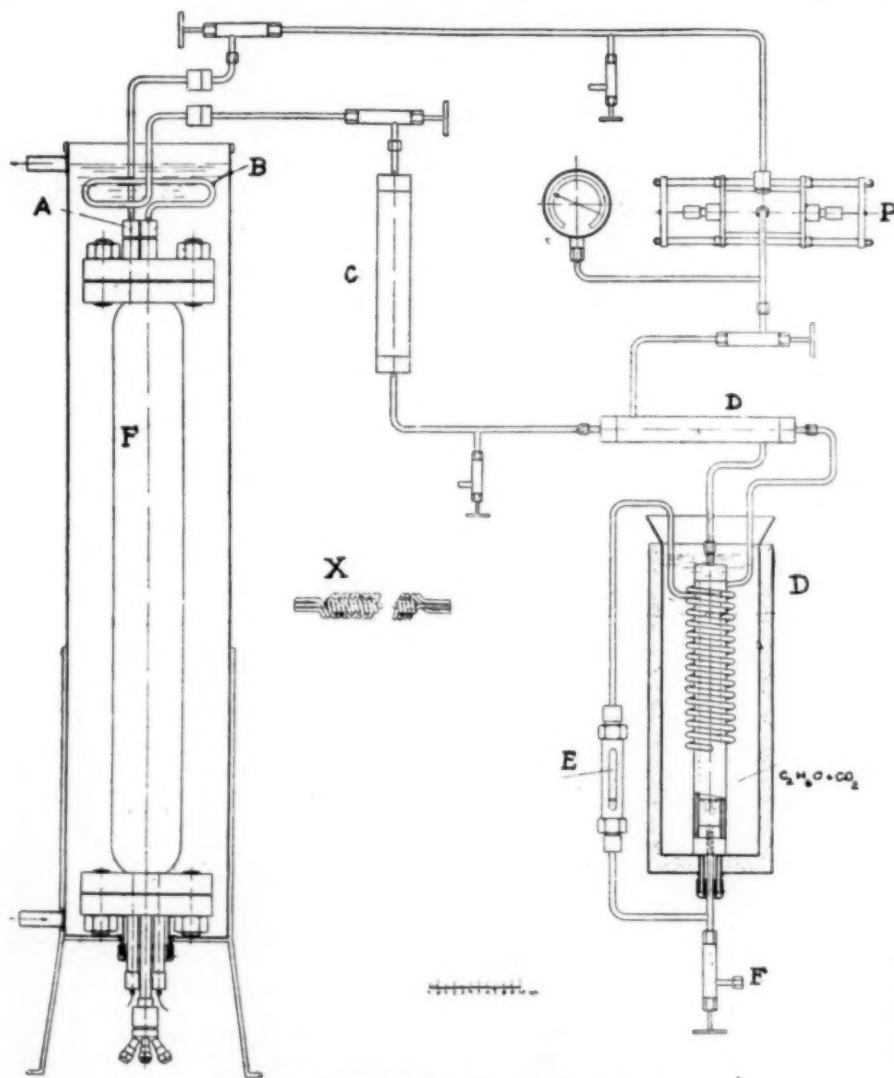


FIG. 4.—ARRANGEMENT WITH GAS CIRCULATION

sisting essentially of a great many capillary steel tubes held together by perforated plates at the two ends.

The catalytic agent was placed into the catalytic chamber in a manner which is acknowledged to be rather primitive. Upon the vertical iron tube which contained the thermoelements, wire-net disks were fixed so as to divide the catalyzer room into a number of chambers one above the other. Some osmium was sprayed into each of these chambers, asbestos being employed to prevent the very fine osmium powder from dropping down through the wire net or from being blown away. The osmium filled only a very small part of the chambers.

Arrangement of Experiments

Two kinds of experiments were made. In the first series of experiments the object was simply to determine the efficiency of different catalytic agents and for this purpose it was sufficient to pass the mixture of hydrogen and nitrogen gases (in the proportion of 3 to 1 by volume) through the furnaces and determine the ammonia that was formed.

with moving gases under high pressure, the authors emphasize that in their work of several years there was never a break of any apparatus or any appreciable trouble.

Catalytic Agents

In investigating the properties of the different elements as to their ability to act as catalytic agents in the formation of ammonia, Professor Haber and Dr. Le Rossignol used the periodic system as the leading idea. It was known that *manganese* and *chromium* are catalyzers. Since in the group containing manganese the elements with highest atomic weights are missing, no advance was possible along this line. On the other hand, in the group containing chromium the two elements with highest atomic weights are tungsten and uranium. In fact the authors obtained a fairly good success with *tungsten* and surprising success with *uranium*.

Uranium may be used in the form in which it is obtained in the electric furnace; it only needs to be roughly crushed. For if it is then slowly heated in the high-pressure gas mixture of one volume gas mixture and three volumes hydrogen, it disintegrates by itself into a fine powder. Commercial metallic uranium, made by Kahlbaum, was used, also commercial metallic uranium, made by de Haen, and uranium made by Moissan's method.

Among the different experiments with uranium, described in the report, is the following in which the furnace, Fig. 2, described before, was used. Uranium was filled into the furnace in form of pieces of pin-head size. The furnace was heated in the saltpeter bath, which also contained an iron tube in which was the thermo-element in porcelain tubes.

An excess pressure of 125 kg per square centimeter was used with a gas mixture of three volumes hydrogen and one volume nitrogen.

Velocity of gas	Temperature	Percentage of NH ₃
32 liters per hour	570 deg. C.	5.65
25.7 liters per hour	505 deg. C.	6.54
9.5 liters per hour	496 deg. C.	9.1
2 liters per hour	503 to 493 deg. C.	11 to 11.9

In the periodic system manganese has as its left-hand neighbor chromium and as its right-hand neighbor iron. Iron is also a catalytic agent for ammonia formation. Below the iron in the same vertical column in the periodic system, we find ruthenium and osmium. Both can be produced in very finely powdered form by similar methods. Ruthenium, however, did not prove a very effective catalytic agent, while *osmium* is very effective. It is notable that in both the iron group and the chromium group of the periodic system the elements with the highest atomic weight—uranium and osmium respectively—are the most effective catalytic agents.

From the different osmium experiments described in the report only the following one may be mentioned, in which a osmium powder layer of 14 mm length and 4.5 mm diameter was used:

Velocity of gas	Pressure	Temperature	Percentage of NH ₃
20 liters per hour	156 atm.	592 deg. C.	4.75
10 liters per hour	156 atm.	572 deg. C.	5.91
1.5 liters per hour	174 atm.	550 deg. C.	7.9
1.5 liters per hour		521 deg. C.	9

The conclusion of the report is an account of a test of longer duration with gas circulation, made to show that the continuous operation of the process with liquification of the ammonia and with the return of the remaining gases into the furnace meets with no difficulties. The heat and cold regenerators proved very efficient.

New Polariscope Tube.—The Bausch & Lomb Optical Company of Rochester, N. Y., has just placed on the market a new polariscope tube, constructed after the design of Frederick Bates, of the U. S. Bureau of Standards. It is a scientifically designed tube suitable for nearly all polariscopic work and especially adapted to sugar polarizations. It has been adopted by the Bureau of Standards.

The Western Metallurgical Field

New Ore Discovery in Colorado

It has been so many years since a promising discovery of ore was made in a new district in Colorado, that the recent announcement of such a discovery in Eagle county has attracted more than usual attention. The scene of the new discovery is on Brush Creek, in the vicinity of the Mount of the Holy Cross, where an earlier discovery of gold and silver ore a number of years ago gave prominence to the camp of Fulford. On account of the difficulties encountered in prospecting and transportation, nothing of importance developed from the earlier discovery; but conditions are now more favorable, and it is probable that the coming spring and summer will find this district thoroughly prospected. At present the snow is too deep for widespread prospecting.

The ore is unlike anything thus far known in Colorado. In appearance it is a grey sandstone, containing minerals of silver which are variously reported to be chloride, bromide and sulphide. Assays indicate the presence of silver in large quantities, several hundred ounces per ton. The richest ore shows bluish spots which, when heated under the blow-pipe or in a muffle, yield metallic silver. A peculiarity of the ore is that some of it contains vanadium in small quantities, say about 1½ to 2 per cent V₂O₅. Should later development show that there is a considerable deposit of this kind of ore, a nice metallurgical problem will be presented to those who ultimately mine and treat it. Such a content of vanadium is not appreciably lower than that of some ores which are treated for this metal alone, and it should not be neglected in this high-grade silver ore. This is a condition which should be recognized before too much of the ore is smelted for the recovery of silver alone, in which event the vanadium would be lost in the slag.

Smuggler-Union vs. Liberty Bell

In June, 1912, the suit of the Smuggler-Union Mining Company against the Liberty Bell Gold Mining Company, both of Telluride, Colo., was tried in the United States district court in Denver. The plaintiff alleged wrongful extraction of ore by the defendant. The result of the first trial was favorable to the plaintiff company which was awarded damages in the sum of \$403,000. The case was appealed by the defendant, and was later heard by the United States Circuit Court of Appeals, sitting in St. Louis. This court affirmed the decision of the lower court on March 3, 1913. It is expected that this decision will be final unless the Liberty Bell company can show error sufficient to carry the case to the United States Supreme Court.

Copper Smelting Resumed at Tooele

During the labor strike in the Bingham district, Utah, last fall, the International Smelting & Refining Company ceased smelting copper at its plant at Tooele, Utah. This action was necessitated by the closing of several mines from which the supply of copper ore was received. Since that time the conditions have improved and are again normal, and the International company is again smelting copper concentrates, with no prospect of further interruption. In the lead smelting department of this plant, four blast furnaces are now in operation.

Consolidated Mercur to Close

For the past two years the annual reports of the Consolidated Mercur Gold Mines Company, Mercur, Utah, have foreshadowed the probable necessity of ceasing operations unless conditions improved materially. The shareholders were asked to authorize the directors to dispose of any or all of the property, which authority was granted. It appears that conditions are not improving and that the famous old mine and mill will be closed within a month or so. This act will close the career of a company which, through its technical staff, contributed greatly to the progress of the cyanide industry, particularly through the invention by George Moore of the vacuum filter.

Notes on Chemistry and Metallurgy in Great Britain

From our Special Correspondent.

Annealing Muffles

At the February meeting of the Birmingham Section of the Institute of Metals Mr. E. H. WALL read a paper on "Annealing Muffles," which dealt with the improvements made in muffles during the last fifteen years or so. The fire-boxes were now made smaller than they were formerly and the flues were arranged to distribute the heat better along the sides and under the beds, while the combustion was more perfect; and as the muffle was enveloped with heat there was an economy in fuel. Cross-fired muffles could be got to work more rapidly than end-fired muffles, and the temperature was maintained with greater regularity. The best arrangement for a muffle was one in which the flame, after passing the bridge and bed, went into a series of graduated portholes opposite to the fire-box; and, after passing these portholes, the waste gases entered a flue under the bed, traveled round the muffle two or three times, according to its width, before making their way into stack. There was still a great deal to be done in improving economical appliances, particularly with regard to gas-firing. At the present time 1.75 cwt. to 2 cwt. of coal was consumed in the gas producer for 1 ton of iron annealed, as against 1 ton of coal to anneal 1 2/3 ton in former times; but even now the theoretical heating was a long way from being attained. If it could be attained 1 ton of coal would heat 36 tons of iron to a welding heat. If the annealing point of copper were put at 1400 deg. Fahr., then 133 heat units would be required to raise 1 pound to that temperature, or 1 ton of coal would be enough to anneal 90 tons of copper. It was not probable that this ideal would ever be attained, but modern muffles had brought down the consumption of fuel by 30 to 50 per cent.

The Institution of Mining and Metallurgy

At a recent meeting of the Institution, papers giving a historical sketch of the Copper Queen Mines at Arizona and dealing with the varieties of ores and methods of treatment, as well as the geology of the Bisbee deposits, the power plants and the reduction works, were presented by Dr. JAMES DOUGLAS, Mr. ARTHUR NOTMAN, Mr. CHARLES LEGRAND and Mr. G. B. LEE.

From Mr. Legrand's paper it appears that the plant at Bisbee consists of five water-tube boilers, with super-heaters and economizers, each of 407 hp, which were fired with California crude oil the thermal value of which was about 18,500 B.t.u. per pound. The pressure of the boilers was 155 lb. per square inch, and the super-heater raised the steam to a temperature of about 440 deg. Fahr. There were three two-stage air compressors, two of which were driven by compound steam engines and delivered about 3500 cubic feet of air per minute, and the third was driven by a triple-expansion, four-cylinder engine and put out 7000 cubic feet per minute. The compressed air at 90 lb. per square inch worked five direct-motion hoists, two geared hoists, some hundreds of air drills and machine tools, some air-jet blowers and some small hoists. Three 500-kw vertical steam turbo-generators supplied three-phase current at 2300 volts, 60 cycles, and part of it was transformed down to 260 volts direct current by two 200-kw rotary converters. Several motor driven hoists were in use and all the pumps were worked electrically, while there were eleven locomotives underground, the largest of which weighed about 7 tons.

At Douglas eight boilers were of 500 hp each, with economizers similar to those at Bisbee; and four were of 520 hp, more than 400 yards distant from the boiler house, and utilized the waste heat from the reverberatory furnaces, working at 180 lb. per square inch to compensate for loss of pressure in the long steam main. There were eleven blowers driven by tandem compound engines, and six of these delivered 300 cubic feet per revolution, and the other five 200 cubic feet per revolution; five blowing engines for converters each delivered

6000 cubic feet of air per minute, and one engine 12,000 cubic feet per minute. There were four 400-kw, d.c., 260-volt generators which drove from 50 to 60 motors, 7 cranes and 14 locomotives; and two 750-kw mixed pressure, turbo-alternates supplied three-phase currents at 2300 volts, 60 periods, which was transformed to 44,000 volts to supply current to a mine 67 miles off.

Mr. G. B. Lee said that the original plant put down in 1902 at the reduction works at Douglas consists of five blast furnaces and four converters, and that plant was duplicated in 1906. The furnaces and converters were arranged on the opposite side of a 60-ft. crane-way, and the re-lining plant was at one end of the main building, which measured 900 ft. x 150 ft. The ores were stored in rock-walled pits, 800 ft. long, 38 ft. wide, and 11 ft. in depth.

Formerly the smelting mixture gave mattes with fully 45 per cent of copper, but gradually changes had been made and the matte was now only from 30 to 35 per cent. A further increase in the proportion of sulphide ores was indicated, and the smelting mixture was becoming finer and less suitable for blast furnaces. The altered conditions necessitated a change in the plant and an enlarged capacity, and consequently six 18-ft. McDougal roasters and two reverberatory melting furnaces, each 91 ft. x 19 1/2 ft. were put down in addition.

Previously the whole of the smelting had been carried on in blast furnaces; and the fineness of the charge together with much alumina in the ores made treatment difficult, while large quantities of dust resulted from the disintegration of the pyrites and the fine concentrates. It was found that the loss of metal in the slag did not amount to the metallurgical loss; and the waste appeared to be due to dust going away with the smoke; but as the volume of the smoke was a million cubic feet per minute, bog filtration was impracticable. Experiment demonstrated that when the smoke was passed through a chamber at 150 feet per minute or less the fine dust settled at short distance, and by introducing wire screens 3 ft. apart an equal settlement was obtained at twice that velocity, and the section of the chambers could be considerably reduced. The screens were so contrived that there were no direct channels through the chamber; and the arrangement was the very opposite of the old system of passing the gases at a high velocity through long flues.

The Iron and Steel Institute

The annual meeting of this Institute will be held in London on May 1 and 2 at the Institution of Mechanical Engineers. The Bessemer medal will be presented to M. Adolphe Greiner, director of the Société Cockerill, of Seraing. Arrangements are in progress for holding the autumn meeting at Brussels.

The Society of Chemical Industry

Two departmental committees of the Home Office were appointed in 1911 to investigate the danger attaching to the use of lead compounds in paints for buildings, vehicles and other purposes; but up to the present they have not presented any report.

Professor H. E. ARMSTRONG and Mr. C. A. KLEIN have recently submitted a paper to the Society of Chemical Industry on "The Behavior of Paints under the Conditions of Practice, with special Reference to the Aspersions cast upon Lead Paints." The authors investigated the contention that illness resulting from living in freshly painted rooms and the bad effects felt by painters were due to volatile compounds of lead; and their experiments led to the conclusion that suspended dust and not lead vapor was the cause; and that earlier experimenters had failed to adequately safeguard against dust being carried forward. With regard to the statement recently made by Professor Baly that noxious organic vapors were emitted during the drying of paints, the authors were satisfied that the ill-effects generally experienced in freshly painted rooms were produced by turpentine vapor, and that symptoms commonly attributed to lead poisoning were simply the effects of turpentine. They were of opinion that the dangers attending the use of lead compounds in paints were merely mechanical dangers of which the chief arose from dust; and that painters ran very

little risk of harm if production of dust were duly prevented.

On the question of the comparative qualities of the materials used as white paints they considered that for outside use, where rain and the impurities always found in town air had access to the painted surfaces, lead paints only were suitable. Zinc paints were not adapted to outside work, but were very serviceable for interior use.

The Institute of Chemistry.

Mr. W. J. A. BUTTERFIELD delivered his second lecture on "Chemistry in Gas-Works" on January 31, when he dealt mainly with impurities, their effects on plant and fittings, and methods of purification. The question of by-products, manufacturing costs, work analyses, efficiency of burners for lighting and heating, further requisite chemical research and the qualifications and training of a gas-works chemist were also considered.

The obstruction caused by the deposition of one hundredth part of a pound of naphthalin in a main was experimentally demonstrated; and the processes employed for the recovery of ammonia, the removal of sulphuretted hydrogen, and cyanogen were described and materials exhibited. In connection with the elimination of sulphur impurities remaining after the removal of hydrogen sulphide, the lecturer referred to the reintroduction in London gas-works of the heated contact-substance process for separating carbon disulphide first proposed by Dr. A. Vernon Harcourt more than 40 years ago. Physical tests of tar and pitch were of little value because their physical properties altered with aging, and the nature and extent of the changes could only be ascertained by chemical analysis. The calorimetry and photometry of gas were demonstrated, and the evolution of the modern 10 c.p. Harcourt pentane standard lamp was illustrated by five successive specimens. The lecturer said that he had made comparative tests of about 150 Harcourt pentane lamps and had never found a greater divergence than 0.2 per cent, except where constructional faults existed. The disturbing effects of variations in atmospheric conditions on photometric measurements were discussed.

Transmutation or Expulsion

The general opinion among practical scientists is that Professor Sir J. J. Thomson has effectually disposed of the supposed conversion of hydrogen into helium and neon by Sir W. Ramsey, Professor N. J. Collie, and Mr. H. Patterson. There can be little, if any doubt about the correctness of Professor Thomson's view that the helium and neon are occluded in and are expelled from the electrodes, and possibly also the glass, because he found that these gases ceased to be evolved after a few days, but were again in evidence when fresh electrodes were substituted. If the transmutation claimants knew of Thomson's results before their papers were made public, it is rather surprising that they should not have withheld their communications until they could establish their case on a firmer foundation.

Engineering Imports and Exports

The returns issued by the Board of Trade for the month of January show a satisfactory all-round improvement in the trade in engineering materials and products as compared with January, 1912—the only decrease being in imports of electrical goods. Iron and steel, including manufactures, were imported to the value of £1,469,352, an increase of £399,759, and were exported to the value of £4,827,729, an increase of £473,491. Imports of other metals, including manufactures, amounted to £2,938,779, an increase of £184,530; and exports reached £1,272,040, an increase of £315,174. Imports of electrical goods totaled £136,192, a decrease of £3,804; while exports touched £368,618, an improvement of £93,527. Imports of machinery were of the value of £673,727, and show an increase of £158,780; and exports went to £3,112,474, an increase of £445,832. Imports of new ships are put at only £1,303, which amount, however, is £1,246 better; and exports rose to £490,425, an increase of £307,203.

Market Prices, February, 1913

Copper opened at £69. The decline in price which has been

continued this month, has followed the same slope from Jan. 14th, dropping rather more sharply about the middle of the month. It was £66 on the 6th, £64 on the 18th, and since rather firmer, closes £64.17.6.

Tin opened at £230 and took a sharp and heavy drop to £220 on the 6th. It then recovered to £224.10 (10th) and fell away more slowly to £217 (18th), recovering to £219 on the 20th, but falling again to £213 by the 24th, touched £220 on the 26th, and closes £218.

Haematite, opened 81/3 and remained steady till the 14th, when it dropped to 80/6 in three days. Kept at this price till the end of the month.

Scotch Pig opened 71/3, and has been exceptionally uneven, being practically the same price on the 11th, and dropping to 65/- by the 18th, closed better at 66/7½.

Cleveland was 65/9 at the end of January and remained about 65/- till 13th, when it dropped away 4/- in five days reaching 60/3 on the 18th. It has since improved slightly, closing 61/-.

Lead opened £16.17.6 and remained firm till mid-month, after rather lower, £16.12.6 on 18th, recovering to £16.17.6 on the 20th, but closing £16.12.6.

	£ s.d.
Aluminium, per ton.....	89. 0.0
Alum, lump, loose, per ton.....	5.10.0
Antimony, per ton.....	36.10.0
Borax, per ton.....	17.10.0
Copper ore, 10 to 25 per cent, per unit.....	11/7½ to 12.1½
Copper sulphate, ton.....	23. 0.0
Carbolic acid, liquid, gal.....	1.5
Caustic soda, 70 per cent, per ton.....	10. 0.0
Ebonite rod, per lb.....	5.3
Hydrochloric acid, per cwt.....	5.0
India rubber, Para fine, per lb.....	4.0½
Mica, in original cases, medium, per lb.....	3/6 to 6.0
Petroleum, Russian spot	8¾
Quicksilver, per bottle.....	7.15.0
Sal. Ammoniac, lump, firsts, delivered U.K. per ton..	44. 0.0
Sulphate of ammonia, per ton.....	14. 5.0
Sulphur, recovered, per ton.....	5.10.0
Shellac, cwt.	4. 5.0
Platinum, oz.	9. 5.0
Tin ore, 70 per cent, per ton.....	143 to 145. 0.0
Zinc, Veille Montagne, f.o.b., Antwerp.....	28.17.6

Differences Since January 31st

Higher	£ s.d.	Lower	£ s.d.
Shellac, cwt.	12.6	Alum, ton	5.0
		Copper ore, unit.....	1.3
		Copper sulphate, ton...	2. 5.0
		Carbolic acid, gal.....	4
		India rubber, lb.....	4
		Tin ore, unit.....	5. 0.0
		Zinc, ton	17.6
		Copper, ton	3. 2.6
		Tin	12. 0.0
		Lead, ton	5.0
		Iron Haematite	9
		Scotch Pig	4.7
		Cleveland	4.3

The new Van Ryn Deep mill on the Witwatersrand in South Africa, will be equipped with 80 stamps of 1900 lb. weight, eight tube mills each 16 ft. 6 in. by 6 ft., sand leaching tanks and a Butters slime filtering plant. The estimated capacity of the stamps is 20 tons per day, or 40,000 tons per month of twenty-five working days.

Mexican smelting plants of the American Smelting & Refining Company, at Matahuala and Velardena were compelled to close for a time during February on account of shortage of fuel, caused by interruption of traffic. The company's other smelters at Monterey, Chihuahua and Aguas Calientes have been running without interruption.

Recent Chemical and Metallurgical Patents

Electric Furnaces

Means for Joining Electrodes.—The managers of the electrode factory of Gebr. Siemens & Company, Lichtenberg, near Berlin, in Germany, EDUARD VIERTEL and HERMANN VIERTEL, have improved the method of joining two tapped electrodes together with true centers. Screw-threads can be made exactly in the electrodes with very great difficulty only and, as a rule, the screw-threaded nipples are made a little smaller than the appertaining screw thread. On account of the play between the nipple and the tapped end, the joined electrodes are frequently not in alignment. To improve this condition the ends of the electrodes are provided with cylindrical, conical or otherwise shaped centering means, for example, projections or recesses. Fig. 1 shows the wrong way of joining electrodes. Fig. 2 shows an annular ring projecting from No. 2 electrode and fitting into the corresponding recess of 1 electrode. In Fig. 3 both electrodes 1 and 2 have recesses, and a centering annulus is constituted by a separate member 4, which exactly fits the two recesses. These means allow to bring the electrodes into exact alignment. (1,049,624, Jan. 7, 1913.)

Carbon Electrode for Electric Furnaces.—Mr. BERTHOLD REDLICH, of the Planiawerke, Ratibor, Germany, has invented a new form of carbon electrode with metallic core. When metal, and especially iron, is inserted into the electrode before burning, it absorbs carbon and becomes brittle. A better contact is obtained by casting molten metal, or metallic alloys, into the hollow spaces provided in the electrode. In doing this care has to be taken to give the metal such a shape that the contraction during solidification does not separate it from the carbon. The drawings in Fig. 4 illustrate without

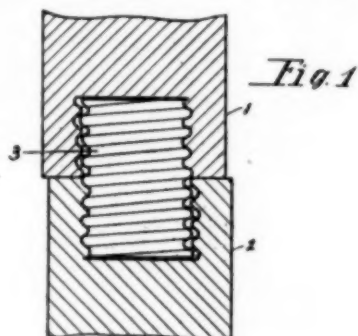


Fig 2

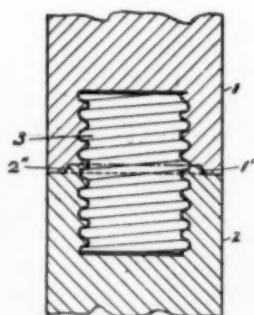
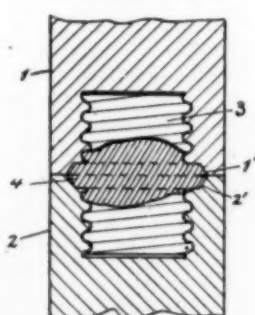


Fig 3



FIGS. 1 TO 3.—METHODS OF JOINING ELECTRODES

further explanation various ways of accomplishing this result. (1,048,581, Dec. 31, 1912.)

Gold and Silver.

Cleaning Filter Fabrics and Screens.—The adhesion or precipitation of solids on filter cloths and screens reduces their efficiency, and necessitates occasional cleaning. An improved method of cleaning these devices is proposed by Mr. MAYNARD T. TROTTER, of Colorado Springs, Colo., the patent being assigned to the Dorr Cyanide Machinery Company, of Denver, Colo.

The process consists in subjecting the surface of the fabric or screen to the action of a sand blast, which, together with the material used, is so regulated that only the heavy exterior coating of the fabric is penetrated. The inventor claims that this process does not cause the deterioration in the fabric such as arises from the use of acid baths; also that the cost of renewals and repairs will be less than with the latter method.

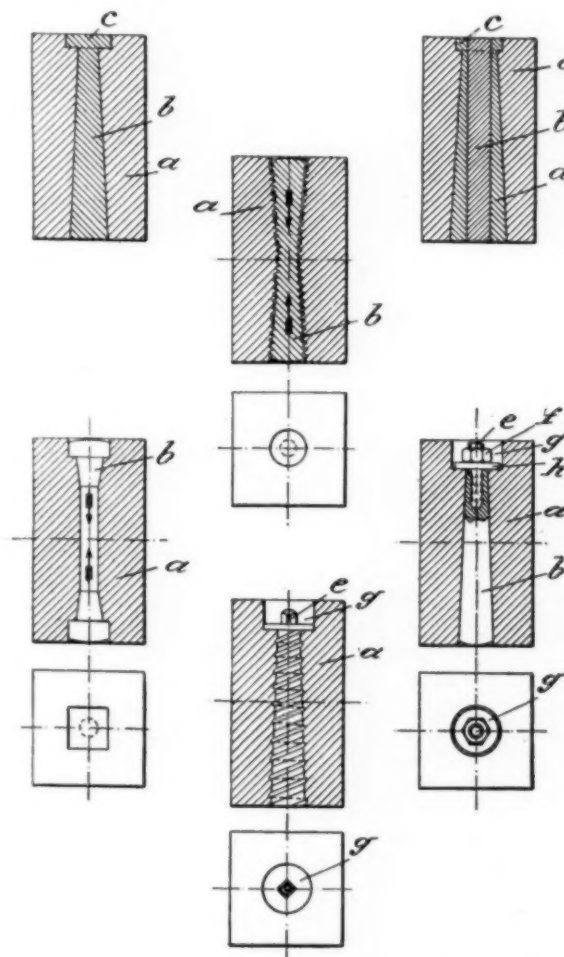


FIG. 4.—VARIOUS FORMS OF CARBON ELECTRODES WITH METALLIC CORE

The method has been demonstrated on screens and filter fabrics in actual use. (1,052,191, Feb. 4, 1913.)

Lead, Zinc and Copper.

Roasting for Magnetic Separation.—The customary method of roasting mixed sulphide ores for subsequent magnetic separation of the iron, consists in freely exposing the ground ore to an oxidizing flame or heat for a period of several hours. In a process proposed by Mr. JAMES B. ETHERINGTON, of Winthrop, Mass., there is a radical departure from the old method, in that the ore is roasted in a practically non-oxidizing atmosphere, at a temperature that is precisely controlled, and for a brief time. All particles of the ore are subjected to the same degree of heat for the same time, and loss by volatilization is avoided. By avoiding a strong draft through the furnace, a minimum of dust is carried out of the stack.

The furnace is shown in Fig. 5. It consists of an unlined cylinder 7, approximately 6 ft. long and 3 ft. in diameter. It is practically surrounded by a hood 41, through which cooling air may be drawn by a fan not shown, but placed above the outlet 42. A hydro-carbon burner 19 is placed at the end of the furnace, through which oil and sufficient air to cause complete combustion are admitted. A drying and preheating cylinder 24 is attached to the furnace, and both have baffles 22 on their interior surfaces for the purpose of causing the ore to fall in

a cascade through the flame. The products of combustion escape by the stack 32, and the roasted ore is discharged through ports 17 adjusted by gates 20. The entire mechanism is supported on rollers and adapted to be rotated by a worm gear.

In operation, the ore is fed through the hopper 30 and travels downward through the furnace owing to its inclination. On reaching the furnace chamber, the particles are subjected to a non-oxidizing heat caused by the blast of oil and air which

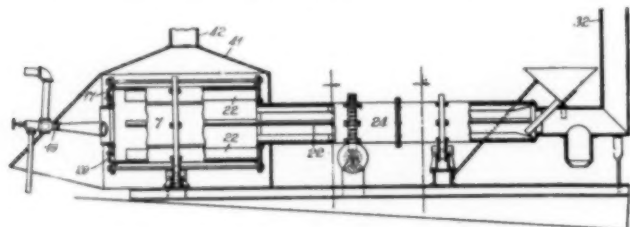


FIG. 5.—FURNACE FOR MAGNETIC ROASTING

is directed onto the wall of the furnace and caused to swirl therein. The rate of feed and size of ports are so adjusted that the ore remains in the furnace for only a few minutes, whereupon it is sufficiently magnetized to enable profitable magnetic separation to be made. The inventor claims that longer time or more oxidizing conditions are not necessary, and that if employed they only cause undue oxidization and volatilization of the constituents of the ore. (1,051,494, and 1,053,486, Jan. 28, and Feb. 18, 1913.)

Method of "Amphidizing" Copper Ores.—In describing the apparatus devised for carrying out his process of rendering the copper in ores soluble, Mr. CHARLES S. BRADLEY, of New York City, gives the following definitions: An "amphigen" is an element which combines with metals to form either an acid or a base, as for example oxygen. An "amphid" is a salt of an acid and a base, each of which contains an "amphigen," as for example copper sulphate, $\text{CuO} \cdot \text{SO}_3$. The apparatus which is used to form the "amphid," is called the "amphidizer."

The inventor proposes by amphidizing to convert the copper contents of ores into soluble form, such as copper sulphate, and then separate the latter from the gangue by solution, later precipitating the copper. His process of amphidizing is continuous in operation. The apparatus consists of a rotary cylinder with feed and discharge at the same end, the former being at the axis of the cylinder and the latter at the periphery. Heat is applied by any suitable burner placed at the end of the cylinder opposite the feed and discharge end. The

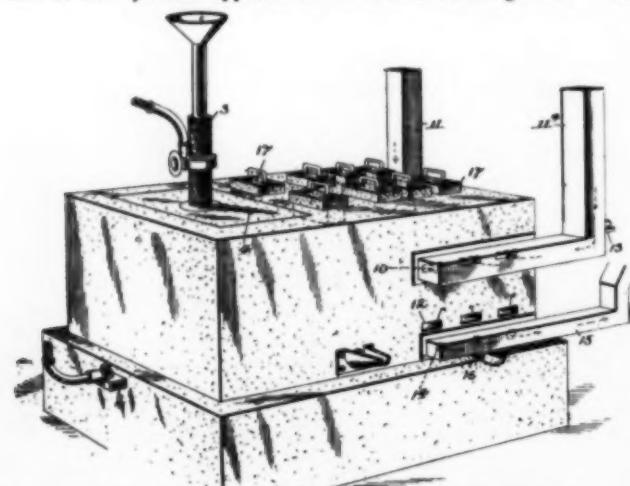


FIG. 6.—PERSPECTIVE VIEW OF ELECTRIC ZINC FURNACE

ore advances through the cylinder by one set of passages and returns by another. This is accomplished by subdividing the reaction space transversely into a number of reaction chambers, the alternate chambers being connected by bridging passages so inclined that the crushed ore advances by gravity as the drum rotates. The intermediate chambers are connected by

bridging passages inclined the other way so that the crushed ore returns by the action of gravity as the drum rotates.

By this arrangement the returning amphidized ore transfers its heat to the incoming ore, and a regenerative effect is produced. This results in an even distribution and regular heat which is so essential causing chemical reactions requiring a definite and steady temperature. In this case the temperature is maintained at about 550 deg. C., or between the dissociation temperatures of iron and copper sulphates. In some cases the heat of reaction will be sufficient to maintain the proper temperature, and it may be necessary to use artificial cooling to prevent the temperature rising too high. (1,052,793, Feb. 11, 1913.)

Electric Zinc Furnace.—In Figs. 6 and 7 are shown two views of a form of electric furnace for the reduction of zinc ores, devised by PETER E. PETERSON, of Butte, Mont. As Mr. Peterson is reported to have made some successful experiments on the electric reduction of the zinc ores of Butte, his furnace is interesting. The furnace consists of a smelting retort 1 in

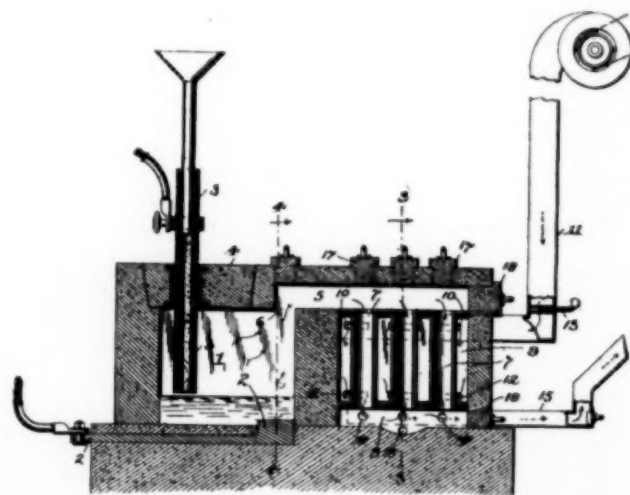


FIG. 7.—LONGITUDINAL SECTIONAL VIEW OF ELECTRIC ZINC FURNACE

which the material under treatment is volatilized by an electric current passing between electrodes 2 and 3. The latter is hollow and surrounds the feed pipe through which ore is introduced into the furnace. The condenser consists of a vapor chamber 5 connected with condensing tubes 7, which extend downwardly to a spelter chamber 8. The tubes are surrounded by a wind box through which cooling air is forced by a fan, entering both sides of the condenser chamber through ducts 11 and 11a. After circulating around the condenser tubes, the air is discharged through ports 12. Uncondensed fumes and vapors are discharged from the condenser through vents 14, leading into the duct 15. The spelter condenses in the tubes 7 and settles into the chamber 8, from which it may be withdrawn through a tap 16. Suitable covers for the condensing tubes are provided as at 17 and 18. It will be noticed that the zinc vapors and cooling medium pass in the same direction, an arrangement which the inventor says has been found highly effective. (1,051,512, Jan. 28, 1913.)

Chemical Industry.

Extracting Chlorine, Iodine and Potassium Hydroxide from Seaweed.—Messrs. FRANK K. CAMERON and RICHARD B. MOORE, of Washington, D. C., and Indianapolis, Ind., respectively, have patented a method of recovering useful products from the ash of seaweed and other marine forms of algae, and have permitted the use of the process by any person in the United States, without the payment of any royalty.

According to one method of treatment, the ash of seaweed is lixiviated with water, the solution drawn off from the solid matter and evaporated to a point where the less soluble salts begin to crystallize. The mother liquor then contains all the soluble iodides and some of the chlorides. It is then elec-

tolyzed in the anode compartment of a diaphragm cell, whereupon chlorine and iodine appear at the anode and potassium hydroxide at the cathode. The chlorine which escapes can be collected, and the iodic acid which is formed can be reduced to iodine by treatment with ferrous sulphate and sulphuric acid.

In a second method, the ash is stirred into water in the anode compartment of a cell. The soluble salts pass into solution, and the same changes occur on electrolysis as described in the first method. (1,051,984, Feb. 4, 1913.)

Iron and Steel.

Manganese Steel.—Mr. HENRY D. HIBBARD, well-known by his research work on impurities in steel, has been granted the following patent for making manganese steel. The object is to utilize manganese steel scrap without material loss of the manganese present in the scrap. If steel scrap containing from 11 to 14% manganese would be charged and worked in the ordinary way in open-hearth furnaces, nearly all the manganese would have to be oxidized before any carbon could be eliminated. This loss can be avoided by melting the manganese steel scrap while it is covered by a bath of decarbonized iron. The latter protects the iron from oxidation while fusion is proceeding. The process can be carried out in an electric furnace. The ordinary charge of an open-hearth furnace can, for instance, be worked down to 0.15% carbon, and the manganese steel scrap thrown into this bath, or the scrap can be heated in the furnace to not more than red heat and then be covered with molten iron of the above carbon contents. When using an electric furnace the latter will be provided with auxiliary heating means. The scrap which is again heated to not more than red heat will be covered with the molten iron, and the melt completed by the application of the electric heating element. When producing manganese steel the small unavoidable loss and the manganese required by the iron have to be added in the form of ferro-manganese. (1,051,840, Jan. 28, 1913.)

Tunnel Furnace.—Mr. ARTHUR RAMEN of Helsingborg, Sweden, applies certain improvements to tunnel furnaces of the canal type, such as are used for the burning of ore briquets at high temperatures and for other similar purposes. The wagons or movable hearths used to carry the material to be treated through these furnaces had a comparatively short life due to the disintegration of the covers and linings of the hearths. This trouble is overcome by providing, in the side walls of the furnace, overhanging portions beneath which the sides of the hearths pass in traveling through the furnace. These overhanging portions are made of hollow tiles and adapted to cooling. The details can be seen from the accompanying drawings and need no description. The tile 14 with a dividing wall 15 and in and outlets 16 and 17 respectively for passing a liquid or other cooling medium through is set up at suitable intervals and bears inclined side walls. By means of these it supports a longitudinal arch in the side wall of the furnace. The tile 19 runs all along the furnace and is cooled in a similar way. (1,051,634, Jan. 28, 1913.)

Process of Improving the Quality of Malleable Iron and Steel.—The late OTTO THALLNER, formerly with Bismarckhütte, in Germany, and later with Electro Stahl, in Reinscheid, worked out shortly before his untimely death a new process for refining steel in an electric furnace on acid hearth. The patent claims that the steel treated according to the process described has a peculiar crystalline grain very similar to, or substantially identical with, the grain which has always been observed in crucible steel, and which apparently gives to such steel its great tenacity. The process may be operated in a hearth-furnace and more particularly in an electric arc furnace, on acid or basic hearth, the heat being furnished by gas, electricity or in any other suitable manner. The feature of the process is the addition to the slag of a metallic oxide capable of giving oxygen to the bath of liquid iron and also to add carbon to such bath. The carbon is best used as carburite, which offers no difficulty in providing carbon both in and below the slag. The amount

of iron ore should not be so much as to cause red-shortness or blowholes in the metal; it will generally be found that between 1 and 2½ per cent of the weight of the liquid metal will be sufficient, while the weight of carbon to be added depends on the amount already in the metal and the finished carbon desired. As an example, it is mentioned that a bath of 5000 kg. desired to be finished at 1 per cent of carbon received from 9 to 15 kg. carbon in the form of carburite, 30 to 35 per cent sand, 30 to 35 per cent lime, 5 to 10 per cent manganese ore to increase fusibility, and 25 to 30 per cent of iron ore. A correspondingly different slag would be made up for a basic hearth. In several chemical equations of an entirely hypothetical character the inventor gives one way of explanation of the reactions which are liable to occur during the refining action. (1,053,454, Feb. 18, 1913.)

Iron-Nickel-Copper Alloy.—Mr. G. H. CLAMER of Philadelphia, Pa., has obtained a patent on replacing part of the nickel of nickel steels with low nickel contents by varying amounts of copper. The latter element is known to raise the elasticity and tensile strength of such nickel steels without diminishing the ductility. Carbon is allowed to be present up to 0.5 per cent. Alloys claimed by this patent contain nickel up to 6 per cent and copper up to 2 per cent. A commercially profitable way of producing these alloys is the use of monel metal or an alloy of 68 per cent nickel, 2 per cent iron and 30 per cent copper. (1,050,342, Jan. 14, 1913.)

Method of Briqueting Flue-dust.—Mr. GEORGE L. COLLARD, of Sharon, Pa., has invented a new method of briqueting the flue dust from blast furnaces or other finely comminuted ore. He mixes limestone, sulphuric acid, and water with the dust until a pasty condition is reached, and molds the plastic mass by hand or any of the ordinary forms of briqueting apparatus. In practice if the dust contains below 0.5 per cent of lime, as is frequently the case, 3 to 4 per cent of crushed or powdered limestone are mixed with it. Sufficient water is then added to moisten the mass and render it suitable for molding. It is then treated with the sulphuric acid which causes the ebullition of the gas and thereby produces the good result of mixing the plastic mass. It keeps the dust in a coherent form until the temperature is raised sufficiently high to melt the metal. (1,047,174, Dec. 17, 1912.)

Process of Oxidizing Ferros to Ferric Solutions.—Messrs. ALEXANDER McKECHNIE and F. G. BEASLEY, of Birmingham, England, have found that the oxidation of ferrous sulphate and chloride solutions into the ferric salts can be rapidly and completely accomplished by subjecting same at a temperature exceeding 100 deg. C. to the action of air under a pressure of 10 atmospheres or upward. The oxidation proceeds so rapidly to completion that large volumes of ferric solutions can be cheaply produced under commercial conditions, whereas the oxidation by atmospheric oxygen or by air in open tanks under normal temperatures and atmospheric pressure proceeds so slowly that the process is practically useless. (1,047,826, Dec. 17, 1912.)

Synopsis of Recent Chemical and Metallurgical Literature.

Gold and Silver

Zinc Wafers for Precipitation of Cyanide Solution.—At the January meeting of the Chemical, Metallurgical and Mining Society of South Africa, Mr. JOHN S. MACARTHUR read a paper dealing with the use of zinc as a precipitant for cyanide solutions, and proposing the substitution of small sheets or wafers of that metal for the customary filament or shaving. The objections to the use of shavings are obvious: the cutting of the filament weakens the fibrous structure of the metal, and causes fractures which are plainly visible under the microscope. These fractures result in a great quantity of "short zinc" and zinc slime which cannot be separated mechanically from the valuable precipitate, necessitating acid treatment. Bullion may contain from 50 per cent to 90 per cent of such material, necessitating elaborate methods of refining. Another

objection to zinc shavings is the difficulty of packing them uniformly in the box, giving rise to spaces of varying resistance to the flow of solution, and consequent imperfect contact and precipitation.

Mr. MacArthur proposes to use wafers of sheet zinc, cut into sizes varying from 1 in. to 2 in. long and $\frac{1}{4}$ in. to $\frac{1}{2}$ in. wide. There is no virtue in any particular size, but oblong pieces are found to pack better in the box than square ones. The resistance of these pieces to the flow of solution is greater than that of shavings, so that boxes must be made with twice the customary fall, and may be only half as deep.

At the Caveira mine in Portugal these wafers have been in use since 1907, and the bullion precipitate is found to contain only about 3 per cent zinc and 12 per cent of other foreign matter. The usual acid treatment is dispensed with. Wafer zinc is used at another mine in Italy and one in Mexico, and the results are said to be excellent. The zinc is found to dissolve evenly, becoming thinner and thinner, until at the end of three or four weeks the wafers disappear altogether without having their structural strength impaired, and without leaving distinguishable debris. The wafers pack in the box in an irregular overlapping fashion, and as their edges are slightly burled or distorted by the cutting process, they do not lie flat on each other, but provide narrow channels for the passage of solution.

A Small Pachuca Agitator for Testing.—In the Colorado School of Mines Magazine for March, 1913, Mr. JOHN GROSS describes a small air agitator which he uses for testing purposes. It consists of a riveted pipe, 24 in. long and 6 in. in diameter, joined to a 7-in. glass funnel. The joint is made with a melted mixture of scrap rubber and sulphur, which is poured into the space between tube and funnel. This mixture hardens quickly, makes a tight joint, but is not brittle. Air is introduced through a piece of glass tubing fitted into the neck of the funnel by means of a rubber stopper, or any other convenient method of holding the tube in place. The lift-pipe is $\frac{3}{4}$ in. in diameter and extends from near the apex of the funnel to within a short distance of the top of the agitator. This lift-pipe is supported in place by suitable means at top and bottom. A settling chamber can be formed at the top of the agitator for the purpose of drawing off clear solution, by inserting a short cylinder, say 4 in. in height and of smaller diameter than the main cylinder, and rising 1 in. above the level of the pulp in the tank. By means of a syphon, clear solution can be decanted as desired.

With slime pulp not thicker than 2:1, the agitation is perfect, and no settling or accumulation occurs in the funnel; but with thicker pulps some thickening and settling occurs. With reground sand this settling does not occur, as the sand seems to have a scouring action which keeps the funnel clear of accumulations.

Compressed air can be obtained from an ordinary tank and pump such as is used for assaying with gasoline. When pumped to 30 lb. pressure the air will last for 20 minutes. The apparatus will serve for the agitation of 32 lb. of dry ore and the necessary solution.

Recovery of Gold by Volatilization.—In the December, 1912, monthly Journal of the Chamber of Mines of Western Australia, Mr. BEN HOWE describes "a process, the application of which, at any rate, is new to the metallurgy of gold." (?) It consists in roasting the ore with salt, volatilizing the gold as chloride, dissolving the latter in a water spray and recovering the gold in a filter press. He states that the gold is suspended "as a black powder" in the water and readily recoverable in the press. Salt is added in quantities varying from 1 to 4 per cent, and the ore is crushed to different degrees of fineness; the finer the ore the quicker the volatilization. A rotary furnace is used, the temperature being maintained at about 1000 deg. C. A higher temperature sinters the ore, and a lower one does not effect volatilization. Laboratory experiments can be made by mixing ore with salt and subjecting the mixture to muffle-heat. The percentage of volatilization is said to be usually over 90 per cent. This process is attracting attention in

Australia on account of its experimental application to refractory ores from the Gwalia Consolidated mines in the East Murchison gold district.

Copper

Notes on Smelting Practice at the Copper Queen Smelter.

—In a paper on the Copper Queen mines and works, Douglas, Arizona, collaborated by members of the technical staff of the company, the portion devoted to the reduction works is contributed by Mr. GEORGE B. LEE. The entire paper is published in *Bulletin* No. 101, Inst. Min. & Met.

The original plant was erected in 1902, and consisted of five blast furnaces and four converters; this was doubled in 1906. The furnaces and converters are arranged in parallel lines on opposite sides of a 60-ft. craneway. The entire plant is on a nearly level site, there being a slope of about $\frac{1}{4}$ per cent from the receiving end to the furnaces. This makes it possible to run trains by easy grades from the ore pits to the charging floor, the distance being about 2000 ft.

The system of bedding ores, which is so common in lead smelters, has not been customary in copper works, where the ores have usually been classified in separate bins from which the components of the smelting mixture were drawn. The bedding system is used at the Copper Queen smelter, all ores being bedded to form a smelting mixture, while fluxes and fuel are kept in separate bins and added to the charges in transit from the ore pits to the furnaces. The ore is measured, but the fuel and fluxes are weighed.

Owing to changes in the ore, causing a reduction in grade of matte from 45 per cent copper to 30 or 35 per cent, and the increase in quantity of fine ore not suited to blast furnace operation, it was necessary to make some changes in the plant. Accordingly, six 18-ft. McDougal roasters and two 91 ft. by 19½ ft. reverberatory furnaces were added. Changes have been made also in the converter practice, substituting basic lined for acid lined converters. The lining of the present converter plant is preliminary only to an entirely new converter plant which will consist of vertical converters, 12 ft. diameter.

The subject of dust losses has been investigated, with the result that the new flues for the calcining and reverberatory furnaces are equipped with wire screens to cause the settling of fine dust. The original experiments to determine the extent of dust loss were made by passing a portion of the furnace gas through a set of closely woven bags. The smoke samples averaged 18,000 lb. each, and the per cent of dust recovered varied from 0.06 per cent to 0.16 per cent, the larger amount corresponding to the larger number of furnaces in operation. The dust contained about 9 per cent copper, 8 per cent lead and 5 per cent zinc. By further experiments it was determined that this dust would settle in flues if the velocity of the gas was reduced to 150 ft. per minute, while by the introduction of wire screens at 3-ft. intervals in the flues, the dust would settle equally well if the velocity were twice as great. The new dust chambers are of novel design, having a steel frame with hollow tile for floor, sides and roof. The latter is suspended from trusses exterior to the chamber. Similar experiments conducted on gas from the converters resulted in the recovery of from 0.015 per cent to 0.035 per cent dust, assaying half as much in copper, but twice as much in lead and zinc as the furnace dust.

As the result of a long series of experiments conducted with a view of devising some suitable way of mixing flue dust with converter slag for subsequent smelting in the blast furnace, the following plan was adopted. The mixing device is a truncated cone set with its axis horizontal. It is mounted on rollers after the manner of a barrel converter, and is revolved at the rate of 5 r.p.m. by a chain and sprocket. Molten converter slag and flue dust are introduced into this revolving cone at its small end; the slag being poured from a ladle and the dust delivered by pipe from a storage tank. The effect is to form a porous, lumpy material which is well suited to blast furnace smelting. The proportions of slag and dust are regulated by one man stationed so that he can see the discharged product which is a guide to the proportions used.

Notes on Copper Smelting in Russia.—The copper smelting works of the Société Minière de Bogoslawsk, Perm, Russia, has been in operation intermittently for the last 150 years, but it is only of late years that modern methods have been employed. A description of present operations is given by Mr. RICHARD DAVEY, in *Bulletin No. 101, Inst. Min. & Met.* The company was one of the first in the Eastern hemisphere to adopt the bessemerizing process, the plant dating back to 1885. The ores consist of chalcopyrite, chalcocite, pyrrhotite, cuprite, malachite and chrysocolla, with a gangue locally known as augito-garnet, with some calcite and a little free silica.

The principal features of the works are sinter-roasting of fine ore and flue dust in pots, granulation of slag, and the concentration of the gold and silver in the converters. The last operation is performed as follows: Three successive charges of matte are made to a converter, the charge being blown to the slagging point after each addition of matte. After the slag is removed from the third charge, the converter contains white metal assaying about 65 per cent copper. This is then blown until the rods used for punching the tuyeres show signs of metallic copper. The converter is then quickly tilted and the white metal run into a ladle and transferred to another converter to be blown to blister for ingot copper. The copper which remains in the first converter is found to contain practically the whole of the gold and most of the silver contained in the original matte, and this is cast into anodes for electrolytic separation.

The blast furnace makes a matte assaying from 30 per cent to 40 per cent copper, and the slags average 0.28 per cent copper. An analysis of the slag shows, in round numbers, 39 per cent silica, 12 per cent alumina, 34 per cent ferrous oxide, 0.3 per cent sulphur, 13 per cent lime and 1.5 per cent magnesia. The sulphur on the charge is 10 per cent, and the quantity of coke used amounts to 7 per cent of the gross charge. The converter slag which is not required for fluxing purposes is added to the furnace forehearth in such a manner that it mixes with the flow of slag and matte from the furnace.

Transformation Point Apparatus

We have just received from the Leeds & Northrup Company a copy of their bulletin describing a new and interesting line of apparatus for the location of the critical points of iron and steel. This bulletin outlines two sets of apparatus, both

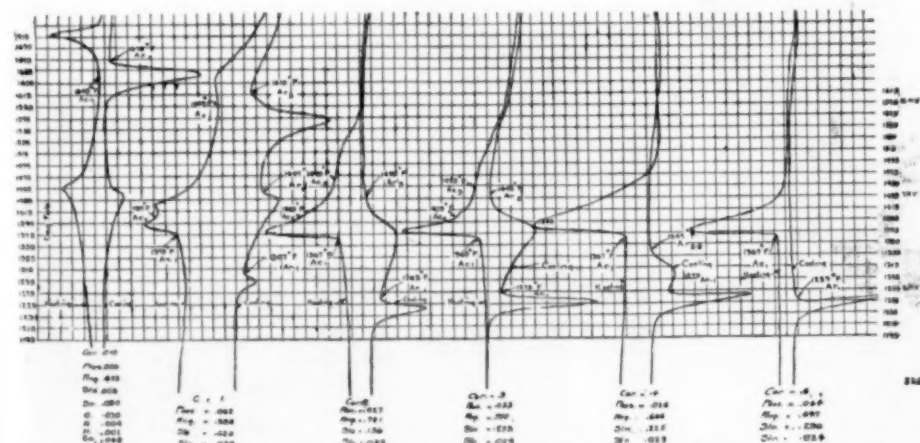


FIG. 1.—HEATING AND COOLING CURVES TAKEN FOR A SERIES OF STEELS WITH TRANSFORMATION POINT APPARATUS

fundamentally the same, but externally very different. We give herewith a reproduction of a series of typical curves taken from this apparatus.

In the curves plotted with this apparatus, the temperature of the sample as it heats and cools is used as one ordinate while the other ordinate is the corresponding temperature difference which exists between the iron or steel and a metallic body

placed in immediate contact with it. The material of this metallic body, or sample holder, as it is termed in the bulletin, is so chosen that throughout the range wherein lie the critical points of iron and steel, the sample holder has no such changes.

Hence the sample and sample holder will heat at the same rate and remain sensibly equal in temperature until the sample undergoes a change involving the absorption or evolution of heat. At this time the temperature difference between the sample and sample holder, previously approximately zero, undergoes a sudden change. This change, even in low carbon steels, is definite and great enough to be readily seen.



FIG. 2.—TRANSFORMATION POINT INDICATOR

only be used with satisfaction to determine cooling curves on account of the effect of fluctuations in furnace. Then, in the second place, the temperature—temperature difference method has the decided advantage of utilizing a quantity (the heat of transformation) whose actual magnitude is small to cause a large percentage change in a second quantity (the temperature difference) which is being subjected to a series of measurements.

To illustrate this last point, suppose a steel to be of such composition as to liberate heat sufficient to change its temperature 2 deg. F., as in cooling it passes 1435. To note by the time-temperature method this, the reading and plotting arrangements must be capable of showing a change of two parts in 1435. On the other hand by the difference method even though the sample and its holder differed initially by as much as 5 deg., a change of 2 deg. would be a change of two parts in five or about 40 per cent.

We reproduce in Fig. 1 heating and cooling curves taken from a series of steels ranging from pure iron to 0.6 carbon steel. The left-hand curve of each pair is the heating curve and the right-hand the cooling curve.

The steel sample should be about 1 in. long by $\frac{3}{4}$ in. in diameter. Where it is required, the holder may be arranged for flat stock.

The temperature and temperature difference measurements are made with thermocouples. The temperature measuring couple is read with a simple potentiometer and the temperature difference is read with a reflecting galvanometer. In one equipment, the Leeds & Northrup potentiometer indicator is used and the readings which are taken practically simultaneously from the potentiometer and from the reflecting galvanometer are afterwards plotted in form of curves.

With the transformation point indicator, an illustration of which is shown in Fig. 2, the observer draws the heating and cooling curve while the process is going on, without taking any readings. In this instrument cross-section paper is wound on a drum which also has wound upon it the slide wire of the temperature measuring potentiometer. The reflecting gal-

vanometer of a difference couple circuit throws its beam upon a movable glass scale which also carries the pen which draws the record. As this galvanometer deflects, the glass scale is moved to keep up with the deflection, thus drawing the pen across the paper.

The galvanometer of the potentiometer circuit is also of the reflecting type and throws its beam upon a fixed scale. This circuit is held at a balance by turning the potentiometer drum, thus drawing the line along the length of the paper. The resultant is of form similar to the curves shown herewith. Both motions are obtained by the rotation of shafts. Thus, at the completion of the test, the observer removes from the drum a finished transformation curve of the steel under test; he has drawn this curve by the simple process of turning two shafts in such a way as to maintain two spots of light on their respective scales. The manufacturers, the Leeds & Northrup Company, state that they have examined many varieties of carbon and alloy steels and that they have yet to find a steel which will not yield definite results on this apparatus.

Recording Differential Pressure Gauges

A comprehensive new line of recording differential pressure gauges has been developed by the Bristol company, of Waterbury, Conn. Some of these recorders have been in successful service continuously since the preliminary models were first sent out in 1908, and the design and construction of the line of these instruments now being placed on the market is based on results obtained in actual service during the last four years. These recording differential pressure gauges are designed for use in connection with venturi meters, Pitot tubes, orifices, combinations of orifices and Pitot tubes, etc., and thereby to record velocities and volumes of air, gas, steam, water and other liquids flowing through mains and pipes. These recorders may also be used to advantage for recording differences and variations of liquid level in steam boilers, pressure tanks, filter beds, process kettles, etc.

Important patents dated September 12, 1911, and September 17, 1912, have been issued to Prof. Wm. H. Bristol, president and founder of the Bristol company, covering novel features resulting from his pioneer work in developing differential recorders in addition to the other numerous lines of recorders which he has originated during the twenty-four years which have elapsed since his first patent was allowed on a recording pressure gauge.

The fundamental principle employed in the construction of this differential pressure gauge is that one pressure is applied to the inside of the operating tube while the other is applied to the outside of the same pressure tube within a closed casing. In order to record the movement of the pressure tube it becomes necessary to transmit its motion to the outside of the pressure tube casing.



FIG. 1.—RECORDING DIFFERENTIAL PRESSURE GAUGE

As the differential pressure to be recorded is usually small as compared with the static pressure, the operative force is correspondingly small, and it is quite evident that it will be impractical to use a stuffing-box around a shaft passing through a pressure casing on account of the friction which would be produced. To avoid the use of a stuffing-box a unique frictionless sealing device is employed as described in detail below.

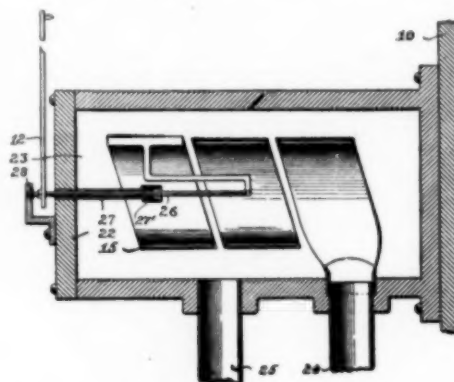


FIG. 2.—INTERIOR OF HELICAL PRESSURE TUBE TYPE OF RECORDING DIFFERENTIAL PRESSURE GAUGE

tionless sealing device is employed as described in detail below.

Fig. 1 shows the exterior of one type of these recorders. Figs. 2 and 3 show part of the interior construction of the spring tube type differential recorder. In Fig. 2 pressure tube 15 is of the hollow helical type used with great success for many years in Bristol recording pressure and vacuum gauges. One of the pressures, the difference of which is to be recorded, is applied to the interior of this tube through the pipe 24 and the other pressure is applied to the exterior of this tube through the pipe 25. It will be noted that the pressure tube 15 is entirely enclosed in the pressure-tight casing 22. The movement of the pressure tube will be in proportion to the difference between these two pressures, and this motion is transmitted to the recording pen arm 12 (outside of the pressure casing) by means of small shaft 26 through the long tubular sleeve 27. The capillary action of the oil or liquid between the sleeve and shaft makes this patented joint both frictionless and pressure-tight.

In Fig. 3, 22 is the pressure-tight casing enclosing the diaphragm pressure tube 14, and in a similar way one pressure communicates with the interior and the other pressure with the exterior of this tube, and its motion is transmitted by means of the rotating shaft through the sleeve 21 to the recording pen arm 12. The length of this sleeve is many times the diameter of the shaft passing through it, differentiating it from an ordinary bearing.

This patented device permits of the recording of extremely small differences between the pressure existing respectively inside and outside of the pressure tube. It has been found that the simple frictionless sealing sleeve through which the pen arm shaft passes does not produce appreciable resistance to the rotation of the shaft, and at the same time capillary attraction and adhesion prevent leakage of even high pressures from the pressure casing.

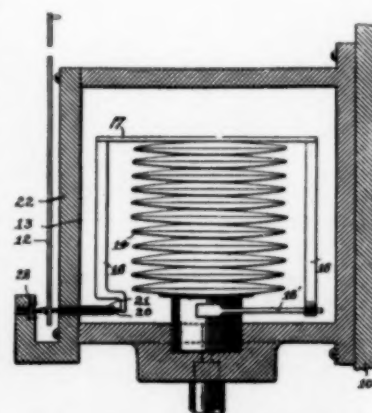


FIG. 3.—INTERIOR OF DIAPHRAGM PRESSURE TUBE TYPE OF RECORDING DIFFERENTIAL PRESSURE GAUGE

In Fig. 4 diaphragm tube 8 is shown in the pressure chamber of the casing 11 directly connected to one end of the rotating shaft passing through the patented pressure seal, and having its other end directly connected to the recording pen arm 9.

This diagram also shows a set of interlocking valves 17, 18 and 23, which constitute a device for adapting recording differential pressure gauges to practical operating conditions. Cross-valve 22 is shown in an open position connecting the two pressure pipes. Valves 17 and 18 in the pressure pipes are shown closed. Both of these valves 17 and 18 can be opened allowing the static pressure from either pipe to be applied simultaneously to the inside and the outside of the pressure tube of actuating mechanism. The interlocking member 21 can then be turned through an angle of 90 deg., thus making it possible to close valve 22 which completes the connections so that the instrument will record the difference of the two pressures.

Fig. 5 shows a patented safety device consisting of a U-shaped tube 30, partly filled with a suitable liquid such as mercury or water, this tube having enlargements 33 and 34, each having sufficient volume to accommodate the quantity of liquid contained in the U-shaped tube. The length of the U tube varies with the range of the gauge so that the greatest possible head of the liquid contained in the U tube corresponds with the total range of differential pressure that the gauge is designed to record. Should the full static pressure by accident be admitted to either side of the differential gauge, the liquid contained in the safety U tube would instantly be forced up into one of the enlargements, thus allowing the static pressure to be applied simultaneously to both inside and outside of the pressure tube and protecting it from being destroyed. There is a great need for these instruments in many different industries and the field of usefulness for these is rapidly growing. Further information may be obtained from the manufacturers.

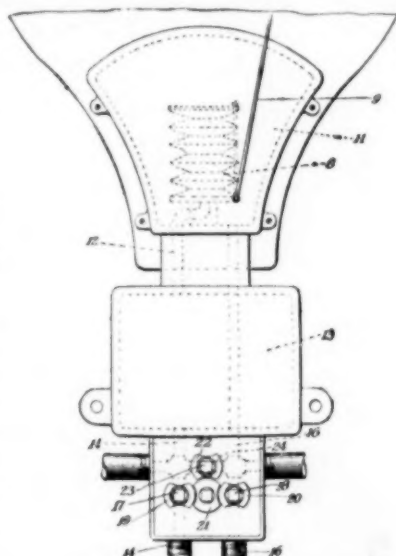


FIG. 4.—CONNECTIONS FOR DIAPHRAGM TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

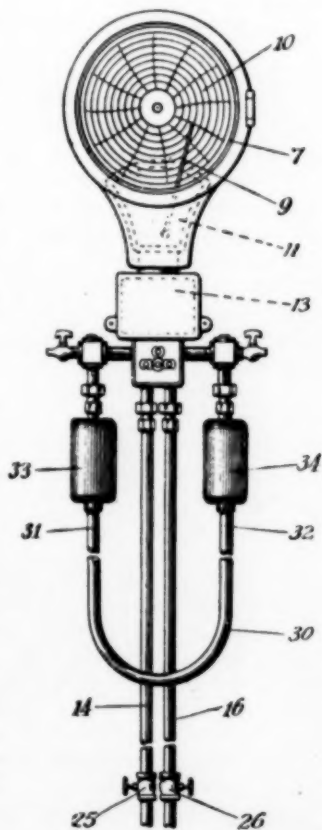


FIG. 5.—PATENT SAFETY DEVICE FOR USE WITH RECORDING DIFFERENTIAL PRESSURE GAUGE

Centrifugal Separation

By Leslie Griscom

The use of centrifugal apparatus in removing liquids from solids and in separating liquids of different densities has a continually widening application. In the textile and sugar industries they have been in very general use for many years. Of more recent application has been the milk-and-cream separator, the medical centrifuge, and the Babcock milk tester. To illustrate the wide scope of the application of the centrifugal machine we will note a few of the materials on which they are used: General laundry work, yarns, piece goods (in rolls), felt boots, rugs, rubber, sugar, hides, curled hair, various kinds of salts, milk, printers' ink, and in bacteriological work, for concentrating germs, crystals, etc., for observation under the microscope.

A form of centrifugal has also been used for extracting honey from the comb, leaving the comb intact, so that it can be filled a number of times by the bees. As a pound of beeswax is said to take several times as much of the energy of the bees as a pound of honey, the honey extractor has proved a great benefit to the bee industry. The amount of moisture left in the material by a centrifugal varies greatly with different types of materials. With certain kinds of chemical salts it will be as low as 2 per cent or 3 per cent, while with fabrics and hair it may reach 50 per cent to 100 per cent of the weight of the dried material, this residual moisture being held by capillary attraction. Possibly few realize the force developed in a centrifugal machine. If we take as an example a 40-in. centrifugal running at 1000 revolutions per minute, a 1 lb. weight at the periphery will exert a centrifugal force of

$$F = 0.000341 WRN^2 =$$

$$0.000341 \times 1\frac{2}{3} \times 1,000,000 = 568 \text{ lb.}$$

in which

W = weight in pounds

R = radius in feet

N² = the square of the number of revolutions.

Taking the same example, if we have a wall of material in the basket 5 in. in thickness the centrifugal head will equal

$$\frac{V^2}{2g} - \frac{V_1^2}{2g} = \left(\frac{40\pi \times 1000}{60 \times 12} \right)^2 - \left(\frac{30\pi \times 1000}{60 \times 12} \right)^2 = 207.2 \text{ ft.}$$

For water this equals a pressure of $207.2 \times 0.433 = 89.7$ lb. per sq. in. at the periphery.

The strain set up in the shell of the empty basket (not allowing for thickness of shell) under the above conditions

$$\text{would be } 0.0000294 V^2 = 0.0000294 \left(\frac{40\pi \times 1000}{12} \right)^2 = 3224 \text{ lb. per}$$

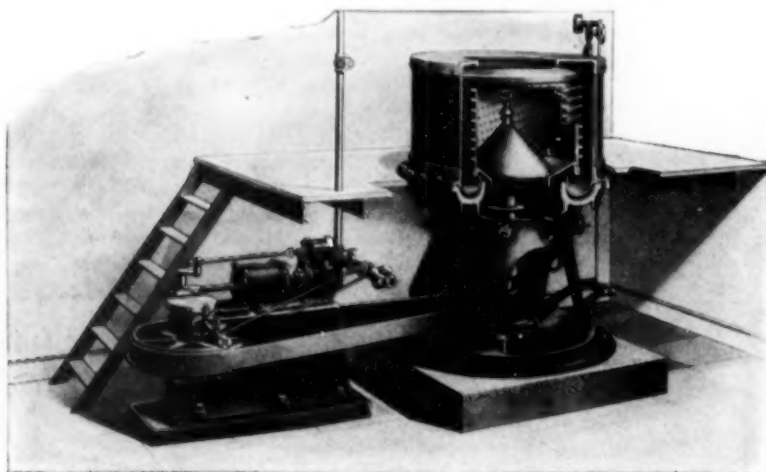
sq. in. cross section of metal.

It is somewhat of a paradox that while a much larger proportion of the liquid can often be removed from chemicals than from fibrous materials the centrifugal machine has had a very general application in the textile trade, while its use for drying chemicals, coal, etc., appears to have been comparatively undeveloped. For this latter purpose a strong, well-designed and well-built machine is necessary to take care of the heavy loads. Further, it must be of a self-balanced type, with bearings well protected, and where acid materials are operated upon the basket, etc., must be constructed of material that will resist corrosion. The machine should be constructed to discharge the material through the bottom of the basket to facilitate emptying. It should preferably be of an under-driven type to give free access to the top of basket. A direct connected motor makes a very convenient drive, doing away with belts and shafting. With the impregnated windings now furnished the motors are able to withstand the corrosive vapors to a remarkable extent. Where there is much explosive vapor a steam engine can be used.

A centrifugal of this type, built by Schaum & Uhlinger, Inc., Second Street and Glenwood Avenue, Philadelphia, has been in successful operation for some years, in the drying of ammonium sulphate and for other purposes.

These machines are substantially built of best material, and are being made to jigs and templates so the parts are interchangeable. The axle steel shafts are ground to gauge. Ample ball thrust bearings are provided which run in a bath of oil. Baskets are furnished either of steel or of material to resist corrosion. Curbs are made of heavy steel plates and lined when necessary to prevent corrosion. These machines are furnished for either belt drive or direct connected to motor through a flexible coupling. They also build a steam engine specially adapted to the operation of machines of this type.

Being under-driven the top of the machine is unobstructed, facilitating the charging and emptying of the basket. On account of the method of construction, and the heavy curb and base, these machines are comparatively free from causing vibrations in the buildings. Where desired, baskets having sloping



CENTRIFUGAL MACHINE

bottoms are provided to facilitate emptying. They also provide a removable top to curb to facilitate removing basket, and provide hand holes in curb to render gutter accessible where the nature of the material handled requires frequent cleansing.

A New Industrial Laboratory and Demonstration Plant

The firm of Mr. Walter E. Lummus, of Boston, Mass., well known by his work in distillation engineering, has erected quite an elaborate industrial laboratory or rather demonstration plant, equipped to operate all kinds of processes involving distillation and evaporation on a manufacturing scale and an outline of some of the interesting work which is being carried out now in this laboratory is worth giving.

They are now building and testing apparatus for coating and drying fabrics with plastic compounds and recovering the solvents; also a plant for producing turpentine and other products from wood; further apparatus for extracting rubber from rubber-producing vines and shrubs.

There is further in course of construction a large continuous still for concentrating wood alcohol with recuperative heater, with a capacity of 10,000 gallons a day.

Nearly all the products of this firm are large apparatus for operation on a commercial scale, but they have under construction several interesting experimental apparatus. Among them is a new outfit for demonstrating fractional distillation, being built for Columbia University, further sundry experimental apparatus to be used, chiefly in connection with wood chemical and solvent extraction problems. Among these experimental apparatus is an extracting and distilling outfit which will be used by an important tropical trading company to de-resinate and concentrate sample lots of rubber.

A recent interesting achievement of Mr. Lummus is the development of remarkable economies in the concentration of extracted products by stage diffusion or extraction without heat

which have made possible the recovery of such waste products as wool grease at an actual profit.

We reserve a more detailed description of the equipment of the new Lummus industrial laboratory for a future issue.

"Semi-Producer" Furnaces

The recent rapid increase in the price of oil as fuel has brought various other processes prominently to the front, especially the use of powdered coal and the use of producer gas.

The W. S. Rockwell Company of New York City, has worked out an interesting solution of some difficulties experienced in the use of producer gas. Hot producer gas for average heavy work usually implies large, expensive flues, which soon become tar and soot clogged, and, in any case, entail the loss of a considerable portion of the sensible heat of the gas

and take up a large amount of room. Cold producer gas means that moderate temperatures only are obtainable unless regeneration or elaborate recuperation is resorted to, since all the sensible heat of the gas has been lost, a portion of the volatile gases condensed to tar and there remains only a lean gas diluted with a large and worse than useless volume of nitrogen. The solution of the W. S. Rockwell Company is very simple; it is to combine the furnace and gas producer; thus making a small, simple, self-contained, easily handled, efficient unit, covering within itself the entire heating operation from coal pile to heated product—having all the advantages of a hot producer gas furnace without the disadvantages of a separate producer and expensive connecting flues.

In this type of furnace the generation and distribution of the hot gas to the material in the furnace is accomplished without the loss of any sensible heat, without the condensation and

consequent loss of any of the tarry gases and with little loss of heat by radiation.

On these principles they have developed a variety of furnaces covering many classes of work. At all stages of the development the fact that good heating conditions must not be sacrificed in order to use a cheap fuel was borne in mind and the result has been a line of furnaces which involve such essential principles of good and economical heating as under-firing, recuperation, down-draft and absolute control of temperature.

While it has been impossible, of course, to cover all sizes and types of furnaces by this combination of furnace and gas producer, yet the fields which have so far been developed for this combination by the W. S. Rockwell Company, cover annealing, hardening, tempering, carbonizing, enameling, and other miscellaneous operations.

Metals and Alloys Free from Carbon Produced by the Thermit Process is the title of a very attractive and well written pamphlet of 30 pages, issued by the Goldschmidt Thermit Company of New York City. It contains concise technical information on the properties and uses of the following metals and alloys: ferrotitanium, chromium, ferrochromium, manganese and manganese alloys, ferromolybdenum, and vanadium.

Titanium in Steel—a Money Saver is the title of a very instructive and profusely illustrated pamphlet of 48 pages, just issued by the Titanium Alloy Manufacturing Company of Niagara Falls, N. Y., on the uses and benefits of ferrocobalt-titanium as a dioxidizer and scavenger of steel for various purposes, including, of course, rail steel. Particularly attractive and instructive are a series of microphotographs and a table giving the results of mechanical tests of steel treated with titanium. The pamphlet contains also the specifications which have been carefully worked out by the Titanium Alloy Mfg. Company to govern the addition of ferrocobalt-titanium for all kinds of steels.

Personal

Mr. **E. A. Austin**, manager of the Guggenheim interests in the Iditarod district, Alaska, was a recent visitor at Stanford University, California, from which institution he graduated in 1906.

Mr. **Herman Bacharach**, until recently located at 722 Lewis Block, Pittsburgh, Pa., has moved into larger quarters at 1009 Hartje Building. He has enlarged the number of his specialties in scientific and industrial instruments, which now include smoke and dust recorders, specific gravity meters for gas and liquid, carbon dioxide recorders and engine indicators with automatic efficiency reckoner, etc.

Mr. **Charles W. Badgley** has resigned his position as chemist in the research laboratory of the American Smelting & Refining Company, at Perth Amboy, N. J., and has accepted the position of chief chemist for the El Paso Smelting Works, El Paso, Texas.

Mr. **George H. Benjamin**, consulting engineer, expert in patent causes, and solicitor of patents, announces the removal of his offices on April 15, 1913, from 45 Broadway to 66 Broadway (Manhattan Life Building) New York City.

Mr. **D. W. Buckby** has been appointed superintendent of the Stewart mill at Wallace, Idaho. For some time past Mr. Buckby has been engaged in experimental work for the Federal Mining & Smelting Company, of Idaho.

Mr. **J. Parke Channing**, consulting engineer for the Miami Copper Company, recently inspected the company's property at Miami, Arizona.

Mr. **E. C. Engelhardt** has left Garfield, Utah, to accept the position of smelter superintendent with the Idaho Smelting & Mining Company, at Clayton, Custer County, Idaho.

Mr. **Le Roy Gordon** has been appointed manager of the New York office of the Nelson Valve Company and Yarnall-Waring Company, with offices at 30 Church Street, New York City. Mr. Irving N. Beeler has been appointed district sales manager for central New York with offices in the Bastable Building, Syracuse, N. Y.

Mr. **Franklin Guitermann**, technical director for the American Smelting & Refining Company, with headquarters in New York City, was in Denver in March on a periodical trip of inspection.

Mr. **A. B. W. Hodges**, who left British Columbia three years ago to assume general management of the Cerro de Pasco company in Peru, has finished his contract and will return to the United States.

Mr. **D. C. Jackling**, general manager for several copper companies in Utah, Arizona and New Mexico, will change his residence from Salt Lake City, Utah, to San Francisco, Cal.

Mr. **J. G. Kirchen** was recently re-elected a director of the Tonopah Merger Company, and will continue as general manager of the company's operations at Tonopah, Nev.

Mr. **W. A. Kunkle** has resigned as superintendent of the Ajax mill at Victor, Colo., and will engage in consulting practice in Denver, Colo.

Mr. **William D. Leonard**, superintendent for the Garfield Smelting Company, Garfield, Utah, was a recent visitor in Denver.

Mr. **Edgar Rickard**, manager of the *Mining Magazine*, London, has been making an extended visit in the United States.

Mr. **Hallet R. Robbins** has been appointed assistant professor of metallurgy at the State College of Washington, Pullman, Washington, succeeding the late R. E. Sampson.

Mr. **J. F. Thorn** has resigned as general superintendent for the Goldfield Consolidated Mines Company, Goldfield, Nev., and will be succeeded by Mr. Albert Burch.

Mr. **William Wraith**, general manager of the International Smelting & Refining Company, Salt Lake City, recently inspected the new electrolytic lead refinery built by his company at East Chicago, Indiana.

Obituary

Dr. **Frederic Schniewind**, the widely known pioneer of the by-product coke oven industry in the United States, died on March 12 in New York City. He was born in Germany 52 years ago and a graduate of the University of Heidelberg, where he received the degree of Ph.D. He came to this country in 1889 and by introducing the Otto-Hoffman coke oven he became the founder of the by-product coke oven industry in this country. At the time of his death he was president of the German-American Coke & Gas Company and its subsidiaries, the United Coke & Gas Company and the Coke & Gas Construction Company.

Mr. **George E. Gunn**, who has been a prominent figure in Western mining and metallurgical circles for the past twenty years, died suddenly of heart disease in his apartments in Hotel Utah, Salt Lake City, Utah, on March 11, 1913. Mr. Gunn was fifty years old at the time of his death. He had been prominently connected with the development of Western porphyry copper mines, having had his attention directed to their possibilities while engaged as field engineer for the American Smelting & Refining Company. He also acted as general manager for the company which developed the large lode copper mines and erected a modern custom copper smelter in the Mason Valley district, Nevada. He was associated with Mr. William B. Thomson in the Gunn-Thomson company which originally developed the Inspiration copper property in Arizona, later sold to the Amalgamated interests. The same company is owner of the Magma copper mines in Arizona. Mr. Gunn was a wealthy bachelor, and member of several clubs in Salt Lake City, on the Western coast, and in the East.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

694,349, March 4, 1902, Henry R. Cassel, of New York, N. Y. Relates to a process for recovering gold from refractory ores, such as sulphides, tellurides, etc. Such ores require roasting to set the gold free; by the present process, it is stated that roasting is avoided. The ore is finely pulverized and mixed with water containing about 3 per cent of salt to form a pulp. This pulp is agitated in an open tank (described in patent 694,350), having a lining of carbon connected as one electrode and carbon-coated stirrers connected as the other; the polarity of the electrodes being reversed at will. The current forms hypochlorites and oxychlorides, which oxidize the sulphur, tellurium, etc., setting free the gold. The gold and oxides of alkaline earth metals are precipitated upon the negative electrode as a black powder, from which it is scoured off by the pulp. The gold precipitate is dissolved by cyanide of potassium, which is finally added, and stirring continued; the gold is recovered from the cyanide solution by the usual method.

694,350, March 4, 1902, Henry R. Cassel, of New York, N. Y. Relates to apparatus for carrying out the process in patent 694,349. Within a tank suitably coated with cement, to prevent leakage, and lined with carbon blocks electrically connected as an electrode, there rotates an insulated metal shaft carrying radially projecting stirring-blades of metal and covered with carbon plates. The shaft and stirrers are electrically connected as the other electrode, and both connected to a suitable source of alternating current; or preferably to a continuous current which is periodically reversed. In operation, the ore, mixed with water containing enough salt to render it conductive, is charged into the vat while the stirrers are rotating. After the gold is set free from the ore, a solvent such as potassium cyanide is added to dissolve it, from which it may be recovered as usual.

698,292, April 22, 1902, Edward D. Kendall, of New York, N. Y.

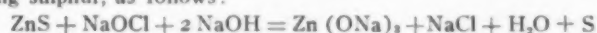
Relates to a process for separating metals from solutions, such as gold from cyanide. Within a suitable tank containing a porous cup is a mass of carbon presenting a large surface, and connected electrically to a source of current at about 15 volts. Within the porous cup is preferably a solution of caustic soda or potash, and a carbon plate connected as the other electrode. The mass of carbon is first connected as cathode and a gold-bearing cyanide solution allowed to flow continuously there-through, the gold being electrolytically deposited upon the carbon. The cyanogen liberated at the anode is absorbed by the caustic soda. When a considerable quantity of gold is deposited, the carbon plate is removed from the porous cup and a silvered copper plate inserted; the cyanide solution in contact with the mass of carbon is now replaced by a stronger cyanide solution and the electric current reversed, making the mass of carbon the anode and the silvered copper the cathode. The deposited gold, etc., is dissolved from the carbon and redeposited upon the silvered copper within the inner porous cup, from which it may be stripped from time to time. After the gold has been removed from the carbon, the strong cyanide solution is drawn off, the silvered copper plate replaced by the carbon plate, the electric current reversed, and the process of treating the gold-bearing cyanide solution repeated.

699,964, May 13, 1902, Frederic H. Long, of Chicago, Ill., assignor to Ross J. Beatty, trustee, of Muncie, Ind.

Relates to an improvement on his earlier patent 650,646. In operating the apparatus described in that patent it was discovered that gases were liberated in the cathode compartment and collected under the diaphragm, thereby increasing the electrical resistance. In the present patent a vent-pipe projects into the cathode compartment to carry off the gases, and also provides means to condense and collect any vapors of acid, etc., entrained by the gases.

700,563, May 20, 1902, Samuel S. Sadtler, of Philadelphia, Pa.

Relates to the extraction of zinc from ores, etc., and from zinc scrap. The ore is pulverized to about 200 mesh and treated with sodium hydrate and hypochlorite of soda in a suitable tank; this dissolves the ore forming zincate of soda and separating sulphur, as follows:



The solution is allowed to settle and the clear liquid run into the cathode compartment of an electrolytic cell, the zinc being deposited upon a suitable cathode. The resulting sodium hydroxide is run into the anode compartment where it may be converted to the hypochlorite by the addition of a suitable quantity of sodium chloride and then used to treat a fresh quantity of zinc sulphide. Zinc scrap, tin scrap, zinc crusts, etc., may also be treated by this process.

700,941, May 27, 1902, Nathaniel Shepard Keith, of Arlington, N. J.

Relates to a process for electrodepositing copper from its solutions, preferably the sulphate. The electrodepositing cells are connected in series both electrically and hydraulically; the electrodes are so proportioned that there is a higher current density per square foot of electrode in the cells at the beginning of the series, where the electrolyte enters than at the end where it leaves. The current density decreases as the copper content decreases, thereby producing a uniform reguline deposit of copper in each cell. The exhausted electrolyte is used to treat a fresh supply of ore and produce more electrolyte.

702,153, June 10, 1902, Jarig Philippus van der Ploeg, of the Hague, Netherlands.

Relates to the extraction of antimony from its ores, etc., and consists in mixing the finely powdered antimony ore with a suitable quantity of quicklime and a monosulphide or polysulphide of calcium or magnesium according to the quantity of antimony present. When water is subsequently added, soluble double sulphides of antimony and calcium or magnesium are formed, whereby all the antimony may be removed from the ore. The solution containing the antimony sulphide is electrolyzed, or may be treated with an acid, such as hydrochloric acid, to precipitate the antimony sulphide.

Book Reviews

The South African Mining Directory. A monthly hand-book. Published by the South African Mining Directory, Ltd., 119 Exploration Bldg., Johannesburg, Transvaal, South Africa. 208 pages, 5¼ x 8; price, £2 10s.

This is a new publication, issued for the first time in November, 1912. Among mining directories it should hold a premier position for accuracy by reason of its frequent publication.

The customary objection to all such directories is that they soon become out of date on account of the changes in the personnel of the business and technical staff of the companies. As this directory is to be issued monthly it will be free from that objection. The first number is quite complete, giving data on the organization of the companies, and a roster of the principal officials. All the mining industries in South Africa and Rhodesia are covered, comprising gold, copper, tin, coal and diamond mines. Statistics on the annual purchases of South African mines, and an alphabetical index of mine officials, complete the volume.

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Refractories and Furnaces. By F. T. Havard, E. M. 6 x 9 in. (15 x 23 cm.), 356 pages, 123 illustrations; price \$4.00. New York: McGraw-Hill Book Company.

The author was in practical work as superintendent of smelting furnaces, and some notes on furnace linings made at that time served as the nucleus of this book. By collecting and adding to these the scattered data on refractory materials, heat conductivity of materials, and heat losses by radiation and convection, the work has been completed.

It discusses refractories primarily and furnace construction incidentally.

The treatment is highly scientific yet intensely practical throughout, and it is just the kind of book which was needed, both for the university and the works. We advise everyone having anything to do, or to work out, in this line, to get the book, and to get it quickly, for while a better book may be written in the future, there is no work at present in print which covers the subject so satisfactorily.

* * *

Electroplating. A Treatise on the Electrodeposition of Metals, including Metal-coloring and Bronzing. By Wm. R. Barclay and Cecil H. Hainsworth. 12mo. (12 x 18 cm.), 399 pages, 62 illustrations. Price \$2.10. London: Edward Arnold. New York: Longmans, Green & Company.

This is primarily a handbook for the practical electroplater, to help him drop rule-of-thumb and to work scientifically. The book is an amplification of lectures delivered at the University of Sheffield by Mr. Barclay. While giving data in ounces Troy and avoirdupois, yet the book lays its main emphasis on using the metric system, and thus tries to lead the electroplaters to the simple, modern decimal system.

The writers not only describe present electroplating methods, but they look ahead and foresee advances in the art. For example, speaking of lead, tin, and antimony, they say that "at the present moment, and writing from an electroplating point of view only, the three metals dealt with in this chapter * * * are of comparatively little interest for the practical worker. It is quite possible and even probable, however, that the immediate future will witness an increase of the commercial possibilities of electroplating with these metals, and some little space should therefore be devoted to an outline of the principal methods of their deposition." Very well said; a book written in such a spirit cannot fail to be valuable and inspiring.

The April meeting of the American Electrochemical Society will show how accurate was the foresight, at least as regards "lead." The field of the electroplater is rapidly widening, as the point of view and education of the electroplater are broadening.